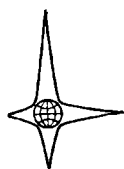Y. MIKHEYEV   I. FAIBISOVICH

THE MINE ELECTRICAL FITTER

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ЭЛЕКТРОСЛЕСАРЬ
УЧАСТКА ШАХТЫ

THE MINE

ELECTRICAL

FITTER

by Y. Mikheyev and I. Faibisovich

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TO THE READER

Peace Publishers would be glad to have your opinion of the translation and the design of this book. Please send your suggestions to 2, Pervy Rizhsky Pereulok, Moscow, U.S.S.R.

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Chapter 1

PROPERTIES OF FERROUS AND NONFERROUS METALS

Iron and steel, known as the ferrous metals, are the basic materials of engineering. However, wide use is also made of nonferrous metals such as copper, aluminium, tin and lead, to mention but a few, and their alloys.

The coal mine electrical fitter has to do with these metals when he services or repairs the mine plant. Therefore, he should have a working knowledge of the physical and mechanical properties of various metals.

1-1. Mechanical Properties of Metals

The mechanical properties of metals are tensile strength, ductility, impact strength, hardness, fatigue strength, wear resistance, etc.

The mechanical properties are determined by means of special testing machines for which test specimens of suitable shape and dimensions are prepared from the respective material.

As to the method of loading, all mechanical tests are divided into:

(1) Static tests, where the load on the specimen is gradually increased during the test, or remains constant over a long interval of time;

(2) Dynamic tests, where the load is applied to the specimen within a very short interval of time, i.e., with a load of impact nature;

(3) Process ability or fabrication tests, where the magnitude of the load applied is immaterial.

In a static tension test, the strength of the specimen is characterized by elastic limit, yield point, and tensile strength. A measure

of the ductility of the specimen is given by its percentage elongation and reduction of area at rupture.

The elastic limit is the greatest stress which can be applied without leaving a permanent deformation after complete release of the load.

The stress may be defined as the load per unit of cross-sectional area of the specimen.

The yield point is the stress at which a marked increase in deformation occurs in the specimen without any noticeable increase in load.

The tensile strength is the maximum stress a specimen will sustain before it breaks. For example, if a steel specimen with a cross-sectional area of 3 cm² is pulled to rupture by a load of 9000 kg, the tensile strength is 9000 : 3 = 3000 kg/cm².

Strength is the ability of a material to withstand the action of an external force without breaking.

All metals change in shape, i.e., deform, under the action of external loads. This deformation may be elastic (temporary) or permanent. Depending on the degree of permanent deformation, metals may be classed as ductile or brittle.

A material in which permanent deformation exceeds elastic deformation is said to be *plastic*; conversely, a material in which permanent deformation prior to its rupture remains small is called *brittle*. The ability of a metal to be deformed easily and permanently under the action of a load is called *plasticity*.

The elongation is the percentage ratio of the increase in length of a specimen prior to rupture to its original length. For example, the length of a specimen to be tested in tension is 200 mm; at rupture, its length reached 280 mm. The per cent elongation therefore is

$$\frac{280-200}{200} \times 100 = 40 \text{ per cent.}$$

The reduction of area is the percentage ratio of the reduced (minimum) cross-sectional area of a specimen at rupture to its original cross-sectional area.

Table 1 gives the main mechanical properties of several metals and alloys determined by static testing.

Referring to Table 1, the first three metals have relatively low tensile strength, but considerable ductility. Copper and aluminium do not possess elasticity, whereas tool steel, though possessing considerable elasticity and tensile strength, has but small ductility. The chromium-nickel steel has the best mechanical properties because it has a high elastic limit and tensile strength coupled with considerable ductility.

Impact (dynamic) testing determines the impact strength of metals.

Table 1

Metal or alloy	Elastic limit, kg/cm²	Tensile strength, kg/cm²	Elonga- tion, %
Iron	1600	3000	30
Red copper	—	2000	40
Aluminium	—	800	30
Unhardened tool steel	3500	9000	10
Unhardened chromium-nickel steel	4000	6000	15
Hardened steel	13,000	18,000	10

Impact strength characterizes the ability of a metal to resist instantaneous loads. In some cases, a metal of adequate strength under static loads may fail, however, under dynamic loads, or impacts. Because of this, in engineering practice many metals must simultaneously possess both a high static-load capacity and a high impact strength. Examples of this are steels used for crankshafts, railway wagon axles, etc.

Impact strength is generally determined by means of pendulum impact testing machines on which a specimen of prescribed shape and dimensions is broken by the impact of a falling pendulum.

Hardness may be defined as the resistance of a metal to local penetration by a harder body.

Hardness measurements are widely used as a method of evaluating the mechanical properties of metals. A definite relationship has been found to exist between the hardness and tensile strength. In some cases a hardness test indicates the strength of a metal. The hardness of materials is determined by forcing a hard steel ball, cone-shaped diamond or pyramid-shaped diamond into the surface of a specimen. This produces an indentation in the surface of the specimen: the harder the material, the smaller the size of the indentation. The most common method is the Brinell hardness test by which a hardened steel ball of known diameter is forced into the surface of the specimen under a predetermined load.

Of great importance to engineering is the ability of metals to resist fatigue failure due to repeated cycles of loading (or vibration) such as occurs in crankshafts, connecting rods, railway wagon axles, valve springs and similar parts. In such parts failure occurs at stresses well below the tensile strength and even the elastic limit. They consequently must be designed and checked for adequate fatigue strength. The factors that greatly influence fatigue strength are the degree of surface finish and the surface hardness. The finer

the surface finish of a part and the harder its surface, the greater its fatigue strength.

The wear resistance of a metal is its ability to resist surface abrasion under dry friction conditions.

For high resistance to wear, many vital machine parts are made of special grades of steel.

1-2. Manufacture of Pig Iron and Steel

Iron almost never occurs in native form. Therefore it is practically always obtained from its ores which are minerals containing iron as oxides (compounds of iron and oxygen). The ores also contain the gangue (all other minerals or waste rock mechanically mixed with the iron-bearing minerals). To win the iron from its ores, the oxides must be reduced to metallic iron by means of carbon, on the one hand, and separated from the gangue, on the other. Both of these processes occur in blast furnaces which produce pig iron.

To be able to separate the gangue or other impurities from the iron, it is necessary to bring them to the liquid state. Since the gangue and other impurities have an extremely high melting point, the latter is lowered by adding fluxes which combine with the gangue and impurities to form low-melting slags. The fluxes charged into blast furnaces together with the iron ore and coke are limestone or quartz rock.

Blast furnaces produce the following grades of pig iron:

(1) Foundry or grey pig iron, intended for the manufacture of castings. This pig iron contains up to 4.5% silicon (which has passed over from the flux or gangue). The carbon in the iron is in the free state in the form of tiny flakes of graphite and gives the iron its grey colour. Grey iron, especially when its phosphorus content is high (up to 0.7%), readily fills moulds during casting and is easy to machine;

(2) Conversion or white pig iron, used for steel making. In this pig iron the carbon is chemically combined with the iron and forms cementite which makes the iron extremely hard and brittle. Such iron shows a light grey (white) fracture;

(3) Special, or ferroalloy pig irons are high in silicon or manganese and are used to make special grades of steel.

An iron having a carbon content greater than 1.7 per cent is called cast iron; if the carbon content is below 1.7 per cent, it is called steel.

Steels may be produced by the converter (Bessemer or Thomas), open-hearth, or electric furnace process.

1. In the Bessemer process molten pig iron is poured into a pear-shaped vessel (the converter) lined with a suitable refractory mate-

rial, and air is blown under pressure at the bottom of the converter and through the molten metal. The blow oxidizes the impurities and the carbon, as well as some of the iron. The heat liberated by the oxidation of the impurities (especially silicon) maintains the temperature in the converter at the level necessary for transforming the pig iron into steel. As soon as the blow is over, the steel must be deoxidized, i.e., the oxygen must be removed. This is done by de-oxidizers (ferroalloys) which are added to the metal. If not deox-idized, steel would be brittle at low temperatures (cold brittleness or shortness) and at high temperatures (red brittleness or shortness)—a condition leading to cracks and tears in rolling. The Bessemer proc-ess uses irons low in sulphur and phosphorus. Bessemer steels are made into rails, galvanized sheet steel, tin plate, and billets for pipe, wire and nail manufacture.

2. The availability of enormous deposits of iron ores high in phos-phorus content has led to introduction of a modified Bessemer proc-ess (known as the basic Bessemer or Thomas process). The modified process eliminates phosphorus from the molten metal by slagging with a flux. The flux is burnt lime which is added to the charge in the converter, whereas in the acid Bessemer process no flux is used. The Thomas converter is lined with dolomite which resists the at-tack of molten slag. After the blow the steel must also be deoxidized.

3. The open-hearth process consists in oxidizing the impurities on the same general principle as in the Bessemer process, but the oxidizing action of the air and gases takes place only on the surface of the charge, the oxygen and metal reacting with each other through the layer of slag floating on the surface of the molten metal.

The charge for an open-hearth furnace may consist of pig iron, purchased scrap, and return scrap from the mills, and steel of almost any desired composition can be obtained. The impurities are re-moved by adding fluxes, for deoxidation ferroalloys are used.

Steels can also be produced in electric furnaces which utilize the heat of the electric arc drawn between the furnace electrodes and the metal charge. Electric arc furnaces produce high-quality steels for castings, alloy steels and nonferrous alloys.

1-3. Carbon Steels

Carbon steel is the general name for alloys of iron and up to 1.7 per cent carbon. Steels may also contain some useful additions (chro-mium, manganese, silicon, tungsten, etc.) and certain harmful im-purities (sulphur, phosphorus).

In the Soviet Union carbon steels are classed as *structural* (mild steels and steels of medium hardness with up to 0.65% carbon) and *tool* (hard steels containing over 0.65% carbon).

Structural steels, in turn, are subdivided into three groups:

(1) Steels furnished according to mechanical properties. The steel manufacturer must guarantee prescribed mechanical properties (tensile strength, yield point and elongation);

(2) Steels furnished according to chemical composition, for which the prescribed per cent content of given elements must be guaranteed.

The above two groups make up ordinary steels.

(3) Steels of improved quality (quality steels) are furnished according to both chemical composition and mechanical properties. These steels are relatively low in impurities and are more uniform in composition.

Carbon tool steels are of two classes: quality and high-quality.

The fundamental property of tool steel is its extreme hardness, attained after special heat treatment. Low-carbon tool steels, due to their toughness, are used for the manufacture of cutter picks, chisels, dies, etc. High-carbon tool steels are made into metal-cutting tools.

1-4. Alloy Steels

Many desired combinations of the mechanical properties in carbon steels cannot be obtained by solely changing their carbon content or by heat treatment. Though it raises the tensile strength and hardness of the steel, an increase in carbon content simultaneously reduces plasticity and toughness. Heat treated cutting tools have high hardness, but are very brittle, cannot withstand impact loading, and lose their cutting edges at high speeds because of temperature rise.

Moreover, carbon steels quickly rust and are easily attacked by acids, bases, and other mediums.

On the other hand, new properties can be imparted to ordinary carbon steels, by adding some elements such as manganese, silicon, nickel, chromium, copper, tungsten, molybdenum, cobalt, etc. These elements are known as alloying additions, and the resultant product as alloy steel.

Alloy steels are classed on the basis of the predominant alloying element into silicon, nickel, chromium-nickel, tungsten, and other steels.

Silicon steels are very elastic and find use in the manufacture of electric machines, transformers and also coil and leaf springs.

Manganese steels (with up to 14% manganese) serve for making machine parts subject to rapid wear because of impact and abrasion, for example, rock-crusher jaws, power-shovel buckets, grinding-mill balls.

Chromium and chromium-nickel stainless steels, in addition to

their high tensile strength, are very resistant to rust and to the action of acids, at both ordinary and elevated temperatures. Chromium-nickel steels are used to manufacture pick boxes and strap links for cutter chains, gears for coalcutting machines, parts for pumps handling acid waters, etc.

The alloying of carbon steel with tungsten, chromium and vanadium produces steels which do not change in microstructure and hardness when heated to temperatures as high as 600°C.

Cutting tools made from these steels can work at extremely high cutting speeds—a characteristic which has given rise to their name: high-speed steels.

1-5. Effect of Carbon and Other Alloying Elements on Properties of Iron and Steel

The properties of iron and steel are mainly governed by the carbon, the most important of the alloying constituents.

The high carbon content of cast iron (as compared with steel) makes the metal fluid in the molten state. Owing to this, it easily fills moulds in casting.

When the carbon is present in graphitic form, as, for example, in foundry (grey) cast iron, the iron has an improved castability, but its mechanical properties are impaired, since the soft flakes of graphite reduce its strength.

Silicon promotes graphitization. Its content is therefore increased in cast irons used for making thin-walled castings.

The presence of manganese in cast iron, on the contrary, retards graphitization and causes the iron to acquire great hardness (known as the chilling effect). Manganese aids in removing the sulphur from the molten iron. However, it produces embrittlement. In foundry iron the manganese content does not exceed 1.3 per cent.

Sulphur considerably lowers the quality of cast iron, especially when the manganese content is low. It decreases the fluidity, makes the iron brittle and liable to cracking.

The sulphur content in coke foundry iron does not exceed 0.05 per cent. Sulphur usually passes over into the iron from the coke during smelting.

The presence of phosphorus in cast iron lowers its mechanical properties, raises its hardness and decreases the impact strength. However, the castability is materially improved, since the iron has a lower melting point and greater fluidity, both of which lead to good mould filling.

When cast iron is alloyed with chromium and nickel in certain proportions, it acquires a greater hardness, a better impact strength, and a higher tensile strength.

As the carbon content of steel increases, the hardness and elasticity increase, but the yield strength and impact resistance decrease, and the steel becomes more difficult to weld. Ordinary steels generally have a carbon content of from 0.05 to 0.63 per cent.

Manganese raises the hardness and tensile strength of steel, and neutralizes the detrimental effects of sulphur. Sulphur is an extremely harmful impurity as it makes the steel brittle at the temperatures of rolling and forging, i.e., between $1000°$ and $1100°C$.

Phosphorus (0.1 to 0.7 per cent) makes steel cold-short, i.e., brittle at low temperatures.

1-6. Hard-facing Alloys

A fairly great variety of materials is available for hard-facing applications. For the purpose of identification they may be classified (according to the form in which they are furnished for use) as follows: cast alloys, powder alloys, and welding-rod alloys.

Cast alloys are mainly cemented or sintered tungsten carbides consisting of tungsten carbides bonded with a solid solution of cobalt, and tungsten-titanium-carbides which are more complex in structure.

In the U.S.S.R. the most widely used of these alloys are BK (VK) sintered carbides furnished as small inserts which can be brazed in the slot of a tool. Either brass or electrolytic copper foil 0.1 mm thick is used as the brazing solder. For brazing, the insert is wrapped with a turn or two of the foil and slipped into the slot where it is held in place with a piece of banding wire. After slightly coating the insert with borax, the tool is heated in a muffle furnace to a temperature of $1050°$ to $1100°C$ until the solder melts, following which the tool is removed from the furnace to force the carbide insert into its slot for a snug fit. The tool is then allowed to cool in dry sand. After cooling, it can be sharpened and dressed on a grinding wheel.

The powder alloys employed for the repair of coal mining plant in the U.S.S.R. are Wocar, Relite and other alloys. One such alloy contains 48 to 59% iron, 8 to 10% carbon, 16 to 20% chromium, 13 to 18% manganese, 3% silicon, and not over 0.5% of each sulphur and phosphorus. The melting point is 1300°C.

Wocar consists of up to 85 per cent powdered tungsten bonded by molasses coke.

Relite contains 95 to 96% tungsten and 3 to 4% carbon. In the U.S.S.R. it is available in three grades: T3 (TZ), K, and 3 (Z). Relite is used to hard-face or hard-surface boring tools.

Prior to applying a powder alloy to a tool cutting face, the face must be cleaned of scale, oil and grease, and a layer of powder alloy

3 to 5 mm thick sprinkled on it. The melting is done with an electric arc. The surface hard-facing thus built up has a thickness of 2 to 3 mm.

Welding-rod alloys are furnished as coated welding rods. The welding-rod coating contains ferroalloys of molybdenum, vanadium and titanium, and also carbides, graphite and chalk. The ingredients are bonded by water glass. The thickness of the hard-facing deposit applied by the arc process does not exceed 2 or 3 mm.

Welding-rod alloys have only found limited application in the Soviet coal industry. In most cases it is Grade T590, which consists of 90% ferrochromium, 5% boron carbide and 5% graphite.

1-7. Properties and Uses of Nonferrous Metals

Nonferrous metals, like iron and steel, also find wide application in engineering, especially the alloys. Among the most widely used are copper, aluminium, lead, tin, zinc, nickel, and magnesium.

Copper is a metal of red colour, ductile at ordinary temperatures, and melting at 1083°C. Pure copper is an excellent conductor of electric current and is therefore the main material with which the electrical fitter has to deal.

Soft (annealed) copper is the material from which cables, conductors, and winding (magnet) wires are manufactured. Semihard drawn copper is used for overhead power line conductors where a higher tensile strength is needed. Hard-drawn copper is used in the form of products such as trolley conductors for mine electric locomotive haulage systems.

In air, copper becomes coated with a thin film of oxide.

Aluminium is one of the light metals finding application in indusstry. Pure aluminium is silvery white in colour, has a melting point of 658°C, is easily worked, and is resistant to a wide number of chemicals. In air, the surface of aluminium oxidizes to form a film which protects the aluminium from further attack.

Aluminium serves as a material for the manufacture of solid and stranded wires, capacitors, switchgear busbars, cable sheaths and conductors, and many light-weight castings. A major disadvantage of aluminium is the difficulty with which it is soldered.

Lead is a soft ductile metal of bluish-grey colour with a melting point of 327.5°C. In damp air, lead becomes coated with a film of oxide. It is used to make protective sheaths for cables, linings for acid-holding tanks and vats, and is used for manufacturing low-melting-point and bearing (babbitt) alloys.

Tin is a soft silver-white coloured metal with a melting point of 232°C. Tin resists the attack of air and weak solutions of acids and bases. It is one of the components of bronzes, low-melting-point and

bearing (babbitt) alloys, and solders, etc. A thin coating of tin reliably protects many metals from surface oxidation.

Zinc is a bluish-white metal with a melting point of 420°C, brittle at low temperatures. In damp air, zinc becomes coated with a film of oxide which protects it against further attack.

Copper-base alloys. Alloys of copper and zinc are called *brasses.* Their mechanical properties differ with the zinc content. Brasses with a zinc content up to 18 or 20 per cent go to make soft ductile products. With up to 40 per cent zinc, brass can be cold forged. Brass is used in the form of sheets, bars, strips and tubes. Many types of electrical contacts are made from brass.

Copper-base alloys containing tin, aluminium, silicon, beryllium and certain other elements are called *bronzes.*

Tin bronzes are harder than brass, have high resistance to corrosion and excellent casting properties. Instead of tin, aluminium has come to be used in bronzes which have a tensile strength not less than that of steel of medium quality. Furthermore. aluminium bronzes are more ductile than steel and have a higher resistance to wear in damp atmospheres.

Aluminium alloys. Of the aluminium alloys special mention should be made of duralumin, an alloy containing copper, manganese, magnesium, silicon and iron. Duralumin is rolled into sheets and special shapes for the aircraft industry, and is shaped into tubes and other products.

Silumin. This is an alloy of aluminium and silicon used for a variety of light-weight castings. The bodies of electric hand drills are cast from this alloy.

1-8. Bearing Alloys

Bearing alloys are applied to the shells or bushes of bearings to form antifriction linings. They must be of low hardness so as not to abrade the shaft and at the same time have sufficient strength to withstand the heavy pressures arising during operation of the machine. These alloys must have a low melting point, adhere well to the shell or bush and also be corrosion resistant. To meet these requirements, any bearing alloy consists of a soft matrix or binder (tin or lead) in which grains of harder constituents are distributed. When a shaft is run-in in a bearing, the grains embed deeper in the softer matrix.

There exist a variety of bearing alloys, babbitt metals being used most frequently.

Babbitt metals are alloys of tin, lead, antimony and copper. Babbitt metals, or simply babbitts, especially the tin-base grades, are very expensive and must be used sparingly.

In many cases, where the load is steady and the speed of rotation is low, nonferrous bearing alloys are replaced by antifriction grey or alloy cast irons.

1-9. Low-melting-point Alloys and Solders

Alloys of tin and lead, bismuth and cadmium have a melting point considerably lower than that of their separate constituents. For instance, an alloy containing 25 per cent lead, 12.5 per cent tin, 50 per cent bismuth and 12.5 per cent cadmium has a melting point of only 60 °C. These alloys are to make fuse links, in automatic fire-protection devices, as type metal, etc.

Solder is the general name given to a class of low-melting-point alloys used to join metals. Their melting points should be less than those of the parts to be joined. Solders may be either soft or hard.

Table 2 gives the tin-lead (or soft) solders most frequently used in the U.S.S.R.

Hard solders have relatively high melting points and may be alloys of copper and zinc (spelter solder), and copper, zinc and silver (silver solders), etc.

Table 2

Solder grade designation	Composition, %			Final melting point, °C	Application
	Tin	Antimony	Lead		
ПОС-90 (POS-90)	89 to 90	0.10 to 0.16	Remainder	222	Soldering of internal seams of brass products
ПОС-40 (POS-40)	39 to 40	1.5 to 2.0	Remainder	235	Soldering of brass, steel and copper
ПОС-30 (POS-30)	29 to 30	1.5 to 2.0	Remainder	250	Soldering of brass, steel, copper, zinc and galvanized steel sheets
ПОС-18 (POS-18)	17 to 18	2.0 to 2.5	Remainder	277	Soldering of lead, brass, steel, copper, galvanized steel; tinning of steel prior to soldering
ПОСС-4-6 (POSS-4-6)	3 to 4	5 to 6	Remainder	265	Soldering of tin plate, brass, copper and lead where seams are lock and rivet jointed

The following grades of spelter solder find application in the U.S.S.R. ПМЦ-36 (PMTs-36), containing $36\pm2\%$ copper (the remainder zinc); ПМЦ-48 (PMTs-48), containing $48\pm2\%$ copper (the remainder zinc); and ПМЦ-54 (PMTs-54), containing $54\pm2\%$ copper (the remainder zinc).

The melting points of the above alloys are between 823° and 888°C. They are used for brazing non-vital parts of brass, bronze, copper, and steel.

ПСр-72 (PSr-72) silver solder contains $72\pm0.5\%$ silver and $28\pm0.5\%$ copper. This solder is used for jointing copper conductors and parts where the joints must have extremely low electrical resistance.

I-10. Corrosion of Metals

Corrosion is a destructive attack on metals which may be chemical or electrochemical in nature.

Direct chemical corrosion is due to corrosive environments such as atmospheric oxygen, various gases and also electrolytes, i.e., aqueous solutions of acids, bases, salts, etc., capable of conducting electric current.

Accordingly, several forms of corrosion are distinguished:

(1) atmospheric corrosion, that due to oxygen in the air;

(2) gas corrosion, that due to some gases;

(3) chemical corrosion, that due to reactions between the metal and gases or liquid dielectrics (nonconductors of electric current) such as petrol, oils, resins, etc.;

(4) electrochemical corrosion, that due to the action of an electric current.

The metals most subject to corrosion are iron and steel.

Between 1890 and 1923 world production of steel and iron was 1766 million tons. During this same period 718 million tons of metal, 40 per cent of world production, was destroyed by corrosion. The loss is very large. This is why corrosion control is of paramount importance.

The corrosion resistance of metals can be improved by adding alloying elements. Small percentages of copper, silicon, nickel, or chromium added to steel retard corrosion appreciably. Sulphur and manganese, on the contrary, promote corrosion.

Corrosion can be controlled by applying oxide films or metal coatings (zinc, lead, copper, tin, nickel, etc.) to the surface. Protection is also provided by coating the surfaces with paints, varnishes or enamels. Smoothly finished or polished surfaces resist corrosion better than rougher surfaces. Because of this, scratches and other surface defects should be removed immediately.

1-11. Heat Treatment of Metals

Heat treatment is an operation or a combination of operations involving the heating and cooling of a metal or an alloy which is in the solid state, for the purpose of obtaining certain desirable conditions or properties due to structural changes.

The heat treatment of steel involves annealing, hardening and tempering (also called drawing).

In annealing, the steel is heated to a temperature ranging from 720° to 930°C, held at this temperature for a certain period of time, and then allowed to cool at a slow rate. Steel, when annealed, becomes more ductile, soft, easier to machine, and can be cold-pressed without developing cracks. The annealing of castings relieves the residual stresses set up by uneven cooling.

Heating is done in annealing furnaces. Cooling may take place either in the same furnace or in the air. When annealing is coupled with cooling in still air it is called normalizing.

Steels are hardened by first heating them to a temperature ranging from 720° to 900°C, holding them at this temperature for a certain length of time, and then quickly cooling (quenching) them. This considerably increases the hardness and strength of the steel. Steels are heated for hardening in heat-treating furnaces which may be of the open-flame fuel-fired, muffle or electric-heated types, in heating baths, and also in a blacksmith's forge when a special furnace is not available. However, heating in a blacksmith's forge cannot ensure uniform quality because the temperature in the forge is always in a state of rapid change and is never the same in different parts of the heating bed.

In flame heating furnaces the hot gases pass over the hearth to heat the parts; in a muffle type furnace the parts are placed in a muffle (chamber) around which the hot gases pass to accomplish heating. Electric furnaces are heated by passing electric current through their heating elements.

A heating bath is a crucible filled with molten lead or salt in which the parts to be hardened are heated. The rapid cooling, or quenching, required for hardening is generally accomplished in a quenching bath, usually water, at a temperature from 20° to 80°C. A greater degree of hardness can be obtained by adding common salt, soda, acids or ammonium chloride to the water. Hardening in oil proceeds at a slower rate. To reduce the brittleness and hardness of a hardened steel, increase its toughness, and also relieve the internal stresses due to the sudden cooling in the quenching bath, it must be tempered or drawn, i.e., reheated after the quench to a temperature not under 700°C, held at this temperature for some time, and then cooled.

Tempering from 180° to 225°C increases the toughness of steel without materially lowering its hardness; tempering from 550° to 650°C gives the steel its best mechanical properties.

Very frequently tempering is carried out without reheating, by utilizing the heat stored by the articles being heat treated.

Where there is no temperature indicating device on hand, the temperature of a heated article taken out of the tempering bath can be determined by the temper colours which appear on its surface due to the formation of an oxide film. As soon as the colour corresponding to the required temperature appears, the article may be quenched in water or oil.

The various temper colours and the corresponding temperatures follow:

Temper colour	Temp. °C	Temper colour	Temp. °C
Light-yellow	220	Violet	285
Dark-yellow	240	Cornflower blue . . .	295
Yellowish-brown . .	255	Light blue	310
Reddish-brown . . .	265	Grey	325
Purple-red	275		

Of late, several methods of surface hardening have been introduced for the heat treatment of steel. Surface hardening is the production of articles having a soft ductile interior and a very hard surface—a property which is of utmost importance for many articles subjected to accelerated wear. This is true of the majority of mining equipment parts.

Surface hardening is mostly accomplished by induction heating whereby high-frequency currents heat the surface layers, and by a gas flame. In both cases the heated surfaces are rapidly cooled as soon as they are taken out of the heating zone.

Carburizing and Nitriding. These two processes belong to surface-impregnation methods which bring about changes in both the structure and composition of the steel. The composition changes due to the impregnation of the surface layer of the heated steel by carbon and nitrogen, respectively. The articles thus treated have a soft ductile core and a very hard surface. For example, gears must be hard at the surface to resist wear and also have tough cores to withstand the shocks they experience during meshing. They are therefore often made of soft steel, and then carburized. After carburizing, the surfaces have a hardness equal to hardened high-carbon steel, but the core metal remains soft.

The impregnation of the surface layer of a soft steel with carbon and the subsequent hardening by quenching is called casehardening. Carburizing involves heating the steel to 890-930°C in contact with

materials high in carbon (carburizers). Such materials may be wood or bone charcoal with potash or soda added, charred leather, potassium cyanide and potassium ferric cyanide. At the end of the carburizing operation the articles are immediately hardened.

In nitriding, nitrogen is introduced into the outer surface of steel parts where it combines with chromium, aluminium, or molybdenum to form a case of hard compounds. The hardness due to nitriding is retained up to 600 °C, while a carburized part, will lose a considerable proportion of its hardness if heated to such a temperature.

Chapter II

INSULATING MATERIALS

Diverse kinds of insulating materials are employed in the manu-facture of electrical machinery and apparatus, erection of power transmission lines and maintenance of electrical plant. If the electrician is to be able to select the right materials for installation and repair work, he must have a working knowledge of their properties.

2-1. Winding (Magnet) Wire Insulations

Wires used for winding coils and for armature conductors, and therefore commonly called winding or magnet wires, are insulated with coverings of fibrous materials (cotton and asbestos yarn, silk, fibrous glass), enamels, or with a combination of an enamel and an outer covering of fibrous material to protect the enamel from me--chanical injury.

In the U.S.S.R., magnet wires are classed according to their insulation by the following grade designations:

ПБО (PBO)—wire covered with a single layer of cotton yarn (single cotton);

ПБД (PBD)—wire covered with two layers of cotton yarn (double cotton). This is the most frequent form of insulation for wires used in motor windings;

ПШО (PShO)—single silk covered wire;

ПЭЛ (PEL)—enamelled wire. The enamel, of the finishing or coating variety, is applied directly to the wire and forms a strong continuous insulating film. Enamelled wire is widely used to wind coils for instruments, relays, transformers and low-power electrical machines. High quality coatings are obtained with the vinyflex type of varnish based on synthetic resins.

ПЭЛБО, ПЭЛБД, ПЭЛШО (PELBO, PELBD, PELShO) are enamelled wires with a fibrous covering (which may be single-cotton, double-cotton, or single-silk).

All the above-mentioned types of insulation are suitable only for relatively low temperatures of up to 105°C.

Where the wire insulation must withstand a higher temperature (of the order of 155°C), those with either asbestos or fibrous glass coverings should be employed; such as ПДА (PDA) wire using a covering of delta-asbestos, or ПСД (PSD) using fibrous glass as the covering.

Delta asbestos insulation consists of asbestos yarn impregnated with a heat-resistant varnish. This type of insulation is used in coalcutter motor windings.

Today many types of electric motors are insulated with silicone varnishes (varnishes based on silicone resins) which can withstand much higher temperatures. The electric motors of coalcutters and cutter-loaders use this type of insulation.

The various electrical insulating materials used in the windings of electric machines, transformers and other apparatus differ in their degree of thermal stability and are grouped into standard classes. These classes are as follows:

Class Y insulation consists of materials such as cotton, silk and paper neither impregnated with nor immersed in a liquid dielectric. Their temperature limit is 90°C.

Class A insulation which consists of materials such as cotton, silk and paper suitably impregnated, or coated, or when immersed in a liquid dielectric. The maximum permissible temperature is 105°C.

Class E insulation consists of synthetic organic films. The maximum permissible temperature is 120°C.

Class B insulation consists of materials such as mica, glass fibre, asbestos, etc., with suitable organic bonding, impregnating or coating substances. The maximum permissible temperature is 130°C.

Class F insulation consists of mica, asbestos and glass fibre with suitable synthetic bonding and impregnating substances. The maximum permissible temperature is 155°C.

Class H insulation consists of materials such as silicone elastomers and combinations of materials such as mica, glass fibre, asbestos, etc., with silicone elastomers bonding, impregnating or coating substances. The maximum permissible temperature is 180°C.

Class C insulation consists of materials such as mica, porcelain, glass, quartz and asbestos with or without an inorganic binder. The maximum permissible temperature is above 180°C.

While ordinary glass is brittle and fragile, fibrous glass is remarkably flexible, elastic and able to withstand heavy mechanical loads.

Glass fibres, while comparable in flexibility with textile fibres, are superior in breaking strength.

Glass fibres are made by drawing molten glass through dies with tiny orifices. The separate fibres are spun into strands and threads which may be woven into cloths and tapes. Glass-fibre yarns can also be used to cover ПСД (PSD) wire.

ПСД (PSD) wire has two lappings of glass yarn. Such insulation has high electric strength, is oil- and moisture-resistant and withstands high temperatures and high mechanical loads. This is the reason why glass-fibre insulation is used where operating conditions are especially adverse (elevated temperatures, ingress of moisture and lubricants).

2-2. Resins, Varnishes, Filling and Sealing Compounds

Resins are organic substances of both natural and synthetic origin which have an amorphous (glass-like and non-crystalline) structure. When heated, resins soften and melt. The most important types of resins are bakelite, shellac and rosin.

Bakelite is a synthetic resin. It is available as stage A bakelite and stage C bakelite. The former dissolves in alcohol and acetone, and fuses when heated to 80°C. The latter is obtained by heating stage A bakelite to 110°-140°C at which it becomes infusible and insoluble in any of the solvents. Stage C bakelite has high strength and excellent insulating properties.

Bakelite finds application as an impregnating agent for wooden parts, and as an impregnating and bonding agent for paper- and fabric-based laminates.

Shellac is a natural resin which is the purified excretions of the lac insect. It is produced in the form of brownish flakes, dissolves in alcohol, has an excellent bonding capacity and finds application as an impregnating agent for windings.

Rosin is a resin obtained from the exudate of pine trees. When dissolved in petroleum oil, it forms various oil-rosin cable filling and impregnating compounds.

Bitumen is a substance close in composition to resin. It can be of "artificial" origin in the form of bitumens obtained as a result of the distillation of petroleum, or of natural fossil origin in the form of bitumens found in the earth and known as asphalts. All bitumens are black in colour, brittle at low temperatures, soluble in benzene and petrol, cannot absorb moisture and are used for preparing varnishes and cable sealing compounds.

Varnishes are solutions of resins or bitumens in oil or volatile solvents or thinners (alcohol, benzene, acetone, etc.). When a coat of

varnish is applied to a surface, it dries either by evaporation of the solvent or thinner, or chemical action, forming a hard lustrous coating which has high insulating capacity and is more or less resistant to air and moisture.

According to the functions they perform, varnishes may be divided into three groups:

(a) impregnating varnishes used to impregnate porous insulation (paper, pressboard, the insulation on the windings of electric machines and apparatus);

(b) finishing varnishes, employed for producing a harder surface over impregnating varnishes, protection against oil, moisture, dirt, etc., and improving appearance. Pigmented finishing varnishes (enamels) are used to obtain insulating films on metal surfaces;

(c) sticking or bonding varnishes used to bond solid insulating materials to each other (as in making micanite from mica flakes), or to a metal or other surface.

In coal-mining electrical equipment, use is frequently made of glyptal varnishes containing glyptal resins (synthetic resins obtained from glycerol, phthalic acid) and drying oils. Glyptal varnishes have excellent bonding capacity, and after baking become very resistant to solvents, and highly resistant to heat.

In recent years wide use has been made in the U.S.S.R. of silicone insulating compounds and varnishes suitable for temperatures of 200°C and higher.

Among them are:

(a) K-43 compound for impregnating electric machine windings operating at up to 180°C;

(b) K-44 varnish for bonding glass-fibre insulation on magnet wires;

(c) K-47 varnish enamel for insulating wires and for bonding glass fibres to wires.

There are also various compounds used to fill cable boxes and to impregnate windings in many types of electrical equipment. As distinct from varnishes, these compounds do not contain a volatile solvent.

Impregnating compounds are used where an insulating material is required to have a higher resistance to moisture than can be obtained with impregnating varnish. Filling (or cable) compounds (termed cable compounds) are used to fill straight-through and tap-off cable jointing boxes and also cable and sealing boxes to protect the cable insulation from moisture.

The principal standard grades of cable filling compounds are:

(a) MK-45—a compound consisting of rosin and a mineral oil; used for filling hermetic joint boxes on cables rated for voltages up to 35 kV;

(b) МБ-70 (MB-70)—a compound consisting of several grades of petroleum bitumen, used for filling cast-iron cable jointing boxes for voltages up to 3 kV and for filling cable end sealing boxes for voltages up to 10 kV when the sealed ends are situated in unheated locations;

(c) МБ-90 (MB-90)—a compound used in the same cases as МБ-70 compound, but only for indoor installations in heated locations.

The numerals 45, 70 and 90 in the above grade designations indicate the softening point in °C of the respective cable compounds.

2-3. Varnished Cloths and Adhesive-coated Tapes

Varnished cloths are insulating materials consisting of a fabric impregnated with a varnish. The insulating value of a varnished cloth comes largely from the varnish with which it is impregnated, and the fabric serves primarily as a support.

Varnished cloths are classed into clear-varnished and black-varnished, according to the varnish used. The first class is obtained by impregnation with a clear oil varnish; the second class, with a black oil-bitumen varnish. Black-varnished cloths have better insulating characteristics, but are less resistant to attack by oil and petrol. They go to insulate machine windings and are available in the U.S.S.R. in two grades: ЛХЧ1 and ЛХЧ2 (LKhCh1 and LKhCh2) which are made in thicknesses of 0.17, 0.20 and 0.24 mm.

Clear-varnished cotton cloth is manufactured in four grades: ЛХ1 and ЛХ2 (LKh1 and LKh2), known as normal grades and available in thicknesses from 0.15 to 0.30 mm; ЛХМ (LKhM), an oil-resistant grade, and ЛХС (LKhS), a special grade.

The limiting or maximum permissible temperature for varnished cloth is 105°C.

For high operating temperatures, use is made of varnished glass cloths. Grade ЛСТЧ (LSTCh) black-varnished glass cloth, impregnated with varnish based on bitumens, drying oils and synthetic resins, can be used where operating temperatures do not exceed 130°C. Grade ЛСК-7 (LSK-7) varnished glass cloth, impregnated with heat-resistant silicone varnishes, permits of application where operating temperatures can reach 180°C.

Electrical circuits and equipment widely employ adhesive-coated or friction tape. It consists of strips of cotton sheeting impregnated with an adhesive insulating compound containing either rubber or bitumen.

Rubber friction tape is made in standard widths of 10, 15, 20 and 50 mm. Bitumen friction tape is available in standard widths of 15, 25, 50 and 70 mm. One of its grades, ЛП (LP) is used to build

up a soft packing between a cable and the throat in a jointing or sealing box.

P.v.c. cable sheaths are jointed and mended with adhesive p.v.c. tape which is manufactured in widths from 15 to 50 mm.

2-4. Moulded Materials—Plastics

Moulded insulating materials are mixtures consisting essentially of various amorphous substances or fillers such as stone and wood flour, and binders such as resins, bitumens, cements, etc. They are shaped in moulds or forming presses. The mixture is first pressed into small lozenges or tablets which are then easily dropped into moulds or press dies. The term "plastics" applies to moulded materials formed from organic compounds. This class includes laminated plastic materials. Laminated materials may be based on paper, fabrics or fibrous glass.

Paper-base laminates consist of sheets of paper bonded with bakelite resin under heat (160°C) and pressure (usually in hydraulic presses). They are made as boards, plates, tubes and rods. These materials possess a relatively high dielectric and mechanical strength; its resistance to arcing is low. An electric discharge over its surface will leave behind a current conductive path of charred (carbonized) material. The maximum permissible temperature for these materials is 115°C. They are known under a variety of trade names (getinax, pertinax, paxolin, etc.).

Fabric-base laminates consist of sheets of cotton cloth bonded with bakelite resin in the same way as paper-base laminates. Fabric-base laminates (known under various trade names as textolite, fabroil, etc.) stand up well to shock loads, have high resistance to wear and good insulation characteristics, but are much more expensive than paper-base laminates.

Synthetic-resin-bonded fibre glass is manufactured from glass cloth sheets bonded with silicone resins under heat and pressure. This material has great mechanical strength and high heat resistance; (the maximum permissible temperature is 185°C).

2-5. Rubber-base Materials

Natural rubber is obtained from the milky sap (latex) of certain plants. Synthetic rubber is produced from alcohol or petroleum products. Pure rubber softens at 50°C, and thus cannot be used as such. Its resistance to heat is improved by vulcanization, i.e., heating with sulphur.

Depending upon the amount of sulphur used for vulcanization, various products will be obtained. An addition of 3 to 10 per cent

sulphur results in a soft rubber, additions of 20 to 50% in hard rubber or ebonite.

Hard rubber or *ebonite* is a solid material which is easy to saw, drill and polish. For electrical engineering purposes, ebonite is produced in the form of sheets, rods and tubes.

In electrical engineering rubber finds use as an insulation for wires and cable conductors. It also is used to make protective articles such as gloves, rubbers, boots, and matting. When heated to 60-75°C, rubber ages, becomes brittle and cracks.

Flexible mine-service cables are made with special sheaths of non-burning synthetic rubber.

Today rubber has many substitutes such as elastic rubber-like p.v.c. base resins. These resins are very resistant to acids, bases and oils, burn poorly and are also insoluble in water. They go to make chemically resistant varnishes, insulation coverings and sheaths for wires and cables.

2-6. Mineral Insulating Materials

Mineral insulating materials are found in the earth's crust and can be used as insulation without heat or chemical treatment. These are mica, marble, slate, and asbestos.

Mica has very high insulating properties, resists elevated temperatures, will not absorb moisture, has considerable mechanical strength and is also flexible. Its applications include insulation for the most vital parts of electric machines and high-voltage installations.

Natural mica is obtained in blocks and splittings. The splittings are bonded together by means of a sticking varnish or resin and built up into sheets, plates and various commercial products which are marketed under various trade names (such as Micanite, Micabond, etc.). Bonded mica products are used as insulating washers, coil insulation, commutator segments and heater plates.

Marble is quarried in the form of blocks which are sawn into boards and polished. Marble can absorb moisture and articles made from it are therefore impregnated with bitumen or oil. It finds use for making switchboard panels, and knife switch and fuse bases. Unfortunately, marble is fragile and non-uniform.

Slate is a laminated mineral which cleaves easily into layers. It finds use as a substitute for marble in small-size switchboards or panels. Slate cannot take a polish, has poorer insulating properties than marble, but can withstand higher temperatures.

Asbestos occurs in rock deposits in the form of veins consisting of tufts of fibres. The latter range from a fraction of a millimetre to several centimetres in length. It can be made into cloths, tapes, asbestos paper and board. The electric insulating capacity of asbestos is

not high; its maximum permissible temperature is 300 to 400°C. Asbestos finds use as a heat insulating material and in the manufacture of asbestos-cement products.

Asbestos-cement is produced by bonding asbestos fibres with cement and press-forming the mixture into board, plate and tube products. Asbestos-cement stands up to shocks, is heat resistant and resists well the effects of electric arcs.

Asbestos-cement boards and plates are extensively used for making multipanel and single-panel switchboards. However, asbestos-cement can absorb considerable amounts of moisture. Therefore it must be well impregnated with linseed oil or bitumen after all drilling and other machining operations have been completed. To do so, the article must first be dried at about 150°C, immersed in oil heated to about 105°C, and allowed to cool together with the oil. After impregnation, the article is allowed to dry in air for twenty four hours. The process is completed by baking it in an oven at 105°C. When an article is impregnated with a bitumen compound, it is dried and then dipped in a bath of molten bitumen. After extraction from the bath and cooling, the excess bitumen is trimmed off with a knife.

Asbestos-cement boards are chiefly used as phase barriers in contactors, controllers and other switchgear.

2-7. Transformer Oil

Oil used in electrical apparatus, known as transformer oil, serves several very important purposes. Above all, it is a reliable insulating medium, a thing of vital importance in high voltage apparatus. Oil, as a liquid, can also easily circulate and act as the coolant for the windings and core of transformers.

In oil circuit breakers oil extinguishes the arc which appears when the circuit breaker is tripped open.

The requirements that transformer oil must satisfy are outlined below.

1. The oil must have high dielectric strength, i.e., the ability to withstand a specified voltage without a breakdown. Transformer oil for electrical apparatus rated for up to 6 kV must withstand a test voltage of 20 kV when in service, and not less than 25 kV when the oil is clean and dry. Oil for apparatus rated for upwards of 6 kV must withstand a test voltage of 25 kV. The test voltage is applied to the testing electrodes spaced 2.5 mm apart.

2. The oil should have a low viscosity for good circulation.

3. The oil must have a high flash point (not less than 140°C).

In coal mines transformer oil may only be used in high-voltage apparatus (transformers and distribution boxes) operating at higher

than 700V. In the case of voltages below 700V, it may only be used in controllers, autotransformers, selenium rectifiers and starting rheostats of stationary plant located in fireproof chambers.

Purification of Oil. During its service, the oil becomes fouled with sludge and impurities such as moisture, insulation fibres, carbon particles, and resinous matter. Such impurities, even in very small quantities, considerably lower the dielectric strength of the oil. It is therefore very important to treat the oil in oil purifiers at regular intervals of time.

Chapter III

FUNDAMENTALS OF ELECTRICAL ENGINEERING

3-1. The Structure of Matter

All things around us consist of very minute particles, atoms and molecules. In all, there are about a hundred different types of atoms. The lightest of them is the hydrogen atom; two hydrogen atoms, combined with an atom of oxygen, form a molecule of water.

The atom of a chemical element is impossible to see, even under the most powerful of microscopes. To gain some idea of the size of an atom, let us imagine some familiar objects enlarged a million times. The head of an ordinary pin would turn into a sphere one kilometre in diameter; a pencil would become 150 to 200 kilometres long and more than 5 kilometres thick. Even then an atom, enlarged a million times, would only acquire a size not greater than a dot on this page.

The atom of any element is very complex in structure. It is composed of a heavy, positively-charged nucleus around which light-weight, negatively-charged electrons revolve. The latter envelop the nucleus to form one or more electron shells. Within each shell, the separate electrons revolve on orbits located at certain distances (energy levels) from the nucleus.

Hydrogen, the simplest of the atoms, has a nucleus with one positive charge of electricity and one electron. Its nucleus has been given the name "proton". The atom of aluminium has a nucleus which consists of 13 protons and 14 particles equal in mass to a proton but carrying no electric charge. Such neutral particles are called neutrons. In the aluminium atom the electron shells contain 13 electrons, the total negative charge of which exactly counterbalances the positive charge of the 13 protons in the nucleus, and the atom remains electrically neutral. The electrons in the aluminium atom are arranged in three levels, with two electrons in the first level nearest to the

nucleus, eight electrons in the second level, and three electrons in the last (outer) level.

Between the positive charge of the nucleus and the negative charge of the electrons exist electrical forces of attraction which are counter-balanced by the centrifugal forces of the electrons due to their rotation round the nucleus. Within the nucleus proper, forces of repulsion act between the positively charged particles. Nevertheless, the nucleus remains stable because its particles are held together by what are called the nuclear forces of attraction.

In metals, the atoms are packed so closely together that their electron shells nearly come in touch. The outer "boundary" electrons, therefore, are subjected not only to the forces of their own nucleus, but also to the forces of the nuclei of adjacent atoms which may break them loose from their atoms. Such electrons are termed free or roaming. They travel between the atoms at random, attaching themselves now to one atom and then to some other atom, and so on.

When a potential is applied across a conductor, the free electrons are caused to travel along the conductor. This flow of electrons in a given direction is called an electric current.

As the free electrons travel, they collide with other atoms of the conductor to cause them to oscillate with an increased amplitude. The greater the oscillations, the higher the temperature of the conductor. At moderate temperatures, the atomic shells behave in an elastic manner and the atoms collide as if they were rubber balls. As the temperature rises, the collisions become more energetic, and some of the outer electrons leave their orbits to reach a level farther away from the nucleus. When an electron is caused to move to a higher level, the atom which absorbs the energy of collision becomes excited. However, this condition does not last very long. The electron again drops to its usual level and the atom loses its excess energy. In doing so, the atom emits what is called a light quantum or photon. The energy of the light quantum is exactly equal to amount of energy given up by the atom.

Thus, each "jump" of an electron towards the nucleus is accompanied by emission of a light quantum. Excited atoms emit light quanta differing in the amount of energy. The human eye is able to detect this difference. Low-energy quanta are sensed as reddish light, quanta of higher energy are seen as orange colours. Violet light quanta carry the greatest amount of energy.

In a body heated to a low temperature the collisions are insufficient in force to excite its atoms and the body cannot radiate any light. As the temperature rises, however, the atoms begin to emit first red-light quanta, then quanta of higher energies. The heated body, therefore, changes in colour.

The atomic nucleus stores enormous amounts of energy. This energy can be utilized in atomic reactors through nuclear reactions. Two types of nuclear reactions are possible. In one type what is known as fission takes place with break-up of a heavy nucleus into light (and simpler) nuclei and liberation of an enormous amount of energy. In the other type, called the fusion reaction, light nuclei combine to form heavy nuclei, again with an enormous amount of energy.

Fission occurs when an atomic bomb is exploded, while fusion is the basis of a hydrogen bomb.

Atomic electric power stations employ the first type of reaction.

Today, in the U.S.S.R. and in many other countries work is going ahead on the industrial application of a controlled fusion reaction.

A. DIRECT CURRENT

3-2. Sources of Electric Current

The simplest source of electric current is a galvanic or primary cell which will continue to supply current until the chemical energy stored in it has been exhausted after which it is necessary to renew the parts of the cell. In its simplest form a primary cell consists of a copper plate *1* and a zinc plate *2* immersed in a solution of sulphuric acid poured in a jar *3* (Fig. 1). The copper plate is found to be the positive pole or electrode (anode), and the zinc plate the negative pole or electrode (cathode). During the action of the cell, the anode wastes away and goes into solution as positively charged ions, leaving the plate electro-negative. The cathode receives positive ions from the solution and becomes electro-positive. If the external circuit attached to the terminals of the cell is now closed, an electric current will flow from the positive pole or terminal, through the external circuit and enter the cell at the negative terminal. In

Fig. *1.* Primary cell

this way chemical energy is directly converted into electrical energy.
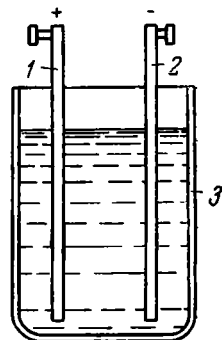
With time the primary cell ceases to operate because of polarization. In most cases the chief cause of polarization is the formation of hydrogen on and about the copper plate which stops the further flow of current. To prevent polarization, the positive electrode is surrounded with a depolarizer, usually a substance containing oxygen. The latter combines with the hydrogen as it evolves at the copper plate, to form water.

2*

In another type of cell (the Leclanché type) the positive electrode is a carbon rod dipped in an aqueous solution of ammonium chloride. The depolarizer is manganese dioxide mixed with graphite. Primary cells are used as a source of current for telephone equipment.

The secondary cell (or storage cell, or accumulator) is another source of electric current.

A storage cell is a device which can be used repeatedly for storing energy at one time in the form of chemical energy for use at another time in the form of electrical energy. Charging a battery consists in connecting the two terminals of the cell to a d.c. supply of proper polarity for a sufficient length of time. Storage cells, usually connected into batteries, may be either of the lead-acid, or the alkaline type. The lead-acid storage battery cell consists of a positive lead plate pasted with an electrochemically active material, and of a negative plate in the form of a lead grid whose pockets are also filled with an active material. Both plates are immersed in a solution of sulphuric acid called the electrolyte. When the cell is charged, the active material of the positive plate oxidizes into lead peroxide, while the active material on the negative plate is reduced to pure sponge lead. When the external circuit connected to the two plates is closed, an electric current flows from the positive plate to the negative plate through the load, and the cell discharges. As the cell discharges the lead peroxide on the positive plate is converted to lead sulphate. The sponge lead on the negative plate is also converted to lead sulphate. In fully discharged cells both plates will be covered with the lead sulphate.

In alkaline storage battery cells the plates are steel grids pasted with an active material. In such batteries the container is of steel and the electrolyte is an aqueous solution of certain bases.

Lead-acid cells have an average working (discharge) voltage of 2 V, while alkaline cells have an average working voltage of about 1.2 V. The number of cells connected into a battery will depend upon the voltage required.

In mines, storage batteries are used as supply sources for electric locomotives and portable safety lamps.

3-3. *Electromotive Force, Terminal Voltage.* *Magnitude of a Current*

A better insight into what occurs in an electric circuit can be obtained from an analogy with the flow of water through a pipe.

Consider two interconnected vessels A and B (Fig. 2). Vessel A is somewhat taller than vessel B. If valve V is now opened, the water will flow from vessel A into vessel B due to the difference in level.

The rate of flow will be the higher, the greater the difference in water level between both vessels.

When the external circuit connected to the plates of a primary or storage cell is closed, a current flows through the circuit due to the "difference in level" of electricity between the two plates. This difference between the electrical states at two points is known as "potential difference" and is the cause of current flow in a closed electric circuit. The potential difference produces what is called an electromotive force (abbreviated to emf). Emf is measured with a *voltmeter*. The practical unit of emf is called the volt (V); 1000 volts is called the kilovolt (kV).

When the water flows from vessel *A* into vessel *B* in our example, some of the water head (difference in level) is lost in overcoming the resistance to the flow due to friction between the water and the pipe walls. In the case of an electric current, too, some of the emf is lost because the external circuit offers resistance to the flow of electric current. Some of the emf is also lost due to the resistance of the internal circuit of the source. The latter portion of the emf is known as the internal voltage drop of the source. What remains is termed the terminal voltage. The terminal voltage is therefore always less than the emf of the source by the internal voltage drop.

Fig. 2. Two communicating water vessels

Referring to our analogy again, the amount of water that can flow during a given interval of time is governed by the water head and the diameter of the pipe. The same applies to an electric circuit. The rate at which electricity can flow through the circuit depends upon the emf and the resistance of the circuit. In electrical terminology the amount of electricity flowing through a circuit during one second is termed the intensity or magnitude of the current. The unit of current magnitude is the ampere; one-thousandth (0.001) of an ampere is called the milliampere (mA). Electric current is measured with an *ammeter*.

Electric energy may also be obtained directly from heat. It is then called thermoelectricity. A thermo-emf can be generated by heating one end of a metal rod. As its temperature rises, the velocity of the free electrons in the metal increases, and they migrate from the heated to the cold end of the rod. The loss of free electrons at the hot end and their increase at the cold end creates a difference in potential between the hot and cold ends of the rod. The resulting thermo-emf is very small in magnitude, being of the order of several thousandths down to several millionths of a volt. This phenomenon is used in engineering for temperature measurements. This is done as follows.
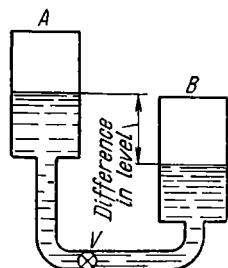
A junction of two dissimilar metals is prepared. The metals may be iron and constantan, copper and constantan, etc., or any others, provided they generate a sufficiently high thermo-emf. Such junctions are called thermocouples. The "hot" end of the junction (or simply the hot junction) is placed at the point where the temperature is to be taken, while the cold junction is connected to a sensitive electrical indicating instrument whose scale is graduated to read directly degrees of temperature. The greater the temperature of the body, the greater the thermo-emf generated, and the greater the deflection of the pointer over the scale.

3-4. Electrical Resistance

Conductors always offer resistance to the flow of electric current. This resistance rises with increasing length and decreasing cross-sectional area of conductors.

The magnitude of electrical resistance is also dependent on the material and temperature of the conductor. For example, a copper wire has a resistance which is considerably lower than that of a steel wire of the same length and cross-sectional area. Any rise in temperature in a conductor is accompanied by an increase in resistance.

The resistance of one metre of a given conductor with a cross-sectional area of one square millimetre at a temperature of $20°C$ is called its specific resistance and is denoted by the Greek letter ρ (rho). The unit of resistance is called the ohm. The resistance of a conductor can be calculated from the formula

$$R = \varrho \frac{l}{s},$$

where R = resistance of conductor, ohms;

ρ = specific resistance of conductor, $\dfrac{ohms \cdot mm^2}{m}$;

l = length of conductor, m;

s = cross-sectional area of conductor, mm².

The cross-sectional area of a round conductor is determined by the formula:

$$s = \frac{3.14d^2}{4} = 0.785d^2, \text{ mm}^2,$$

where d is the diameter of the conductor in mm.

The average specific resistance of several metals is given below:

Copper	0.0175
Aluminium	0.029
Steel	0.13
Nichrome	1.1

The unit of resistance is called the *ohm*, and is the resistance offered to the flow of an unvarying current by a column of mercury of 106.3 mm height and 1 sq mm cross-section at a temperature of 0°C.

Example 1. Calculate the resistance of a copper cable conductor having a cross-sectional area (size) of 25 mm² and a length of 200 m.

$$R = \varrho \frac{l}{s} = \frac{0.0175 \times 200}{25} = 0.14 \text{ ohm.}$$

Example 2. Find the resistance of a steel conductor with a diameter of 4 mm and a length of 50 m.
Cross-sectional area of conductor:

$$s = \frac{3.14d^2}{4} = \frac{3.14 \times (4)^2}{4} = 12.56 \text{ mm}^2.$$

Resistance of conductor:

$$R = \varrho \frac{l}{s} = \frac{0.13 \times 50}{12.56} = 0.57 \text{ ohm.}$$

The megohm is a multiple unit of resistance and is equal to 1,000,000 ohms.

Conductors and Insulators. The resistance of some materials to the flow of an electric current is so high that, when an ordinary voltage is impressed, little or no current will flow through them.

Materials through which an electric current cannot flow are called insulators (or dielectrics). Dielectrics include rubber, porcelain, glass, ebonite (hard rubber), built-up mica (micanite), presspahn (electrical pressboard), mica, chemically-pure water, etc.

Materials which offer little resistance to the flow of electric current are called conductors. All metals are good conductors of electric current.

3-5. Ohm's Law

Fig. 3 shows an elementary electric circuit which consists of an electric energy source *1* (which may be a primary cell, storage battery, generator), an energy converter or load *2* (an electric lamp, heating appliance, motor), and two conductors *3* connecting the load to the source. When the circuit is closed (or completed), a current flows around it. The magnitude of the current depends on the value of the emf and total resistance of the circuit. The three values, i.e., current, emf and resistance, are related by 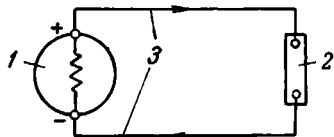Ohm's Law which states that *the current flowing in an electric circuit is directly proportional to the applied emf and inversely proportional to the resistance of the circuit,*

Fig. 3. Elementary electric circuit

or mathematically,

$$I = \frac{E}{R+r},$$

where I = magnitude of the current (or simply current), A;
E = emf, V;
R = resistance of external circuit, ohms;
r = resistance of energy source, ohms.

Ohm's Law is applicable to not only the whole circuit, but also to any part of it.

In the latter case the emf and resistance in the formula must be replaced by the voltage across, and the resistance of, the part in question. Ohm's Law for a part of a circuit may then be stated thus: the current flowing in a part of an electric circuit is directly proportional to the voltage impressed across it and is inversely proportional to the resistance of the part of the circuit, or:

$$I = \frac{V}{R},$$

where I = current, A;
V = applied voltage, V;
R = resistance, ohms.

By multiplying both sides of the equation

$$I = \frac{E}{R+r},$$

by $(R+r)$ we obtain

$$I(R+r) = E,$$

(that is, the emf of a circuit is equal to the current multiplied by total resistance of the circuit) and if we divide both sides of the above equation by I,

$$R+r = \frac{E}{I},$$

(that is, the resistance of the entire circuit is equal to the applied emf divided by the current).

The expression $I(R+r)$ can be rewritten thus:

$$IR + Ir = E.$$

The product IR is termed the terminal voltage of an energy source and is usually denoted by the letter V. It is the voltage drop across the external circuit when a current I flows around it.

The product Ir is called the internal voltage drop. It is denoted by the small letter e and shows how much of the emf is lost within the source when a current I flows through it.

Thus

$$E = V + e,$$

that is, the emf of a source is equal to the voltage across its terminals plus the internal voltage drop.

Example 3. A d. c. generator with an internal resistance $r=0.04$ ohm has a terminal voltage of $V=110$ V when its load current $I=50$ A. Determine the emf, E, of the generator and the resistance, R, of the external circuit.

The voltage drop within the generator:

$$e = Ir = 50 \times 0.04 = 2 \text{ V};$$

the generator emf E then is

$$E = V + e = 110 + 2 = 112 \text{ V}.$$

The resistance of the external circuit is

$$R = \frac{V}{I} = \frac{110}{50} = 2.2 \text{ ohms}.$$

Example 4. An electric circuit consists of lamp and a rheostat connected in series. The lamp takes a current of 1.8 A at rated voltage 115 V. Determine the resistance of the rheostat if the supply voltage is 220 V.

The voltage drop across the rheostat must be

$$220 - 115 = 105 \text{ V}.$$

The resistance of the rheostat must then be

$$R = \frac{V}{I} = \frac{105}{1.8} = 58.3 \text{ ohms}.$$

3-6. Combinations of Loads

Units of utilization equipment, or loads, may be connected in any of two fundamental ways.

Parallel Connection. If several loads such as electric lamps are connected as shown in Fig. 4a they are said to be connected in parallel.
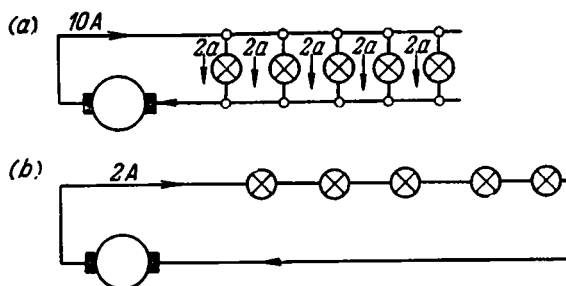

Fig. 4. Combinations of loads:
a—in parallel; *b*—in series

Kirchhoff of Germany discovered that the sum of all the currents flowing towards any point of junction of several conductors is equal to the sum of all the currents flowing away from the same junction point. This is known as Kirchhoff's First Law.

If each of the lamps in Fig. 4a draws a current of 2 A, the total current in the main circuit will be $2 \times 5 = 10$ A. The greater the number of lamps connected in parallel the larger the current flowing in the conductors from the source. When identical loads are connected in parallel, the resistance of the external circuit will be equal to the resistance of one load divided by the number of the loads in the circuit.

Series Connection. When arranged as shown in Fig. 4b, loads are said to be connected in series. In this case the resistance of the circuit is equal to the sum of the resistances of all the loads.

Example 5. Determine the total external resistance of an electric circuit consisting of 10 lamps connected in parallel when the resistance of each lamp is $R_l = 400$ ohms.

$$R = \frac{R_l}{n} = \frac{400}{10} = 40 \text{ ohms,}$$

where n is the number of lamps connected in parallel.

Example 6. Determine the total resistance of a circuit made up of three lamps connected in series. Each lamp has a resistance $R = 75$ ohms. The resistance of the conductors is $r_{cond} = 1$ ohm, that of the source of emf is $r = 0.2$ ohm.

The total resistance of the circuit is

$$R_{total} = 3R_l + r_{cond} + r = 3 \times 75 + 1 + 0.2 = 226.2 \text{ ohms.}$$

In practical circuits loads are very frequently connected in series-parallel, a combination of series and parallel connections.

3-7. Heating of Conductors

Lenz of Russia and Joule of Britain established that the current flowing in a conductor develops heat and that the amount of heat evolved is proportional to the square of the current, the resistance of the conductor and the time the current flows. This relationship, ordinarily known as Joule's Law, is expressed thus

$$H = 0.24 I^2 Rt \text{ cal,}$$

where H = quantity of heat evolved, cal;
I = current, A;
R = resistance of conductor, ohms;
t = time during which current flows, sec;
0.24 = coefficient of proportionality.

Example 7. Determine the amount of heat developed by an electric furnace during 5 hours if the furnace takes 5 A at 120 V.

The resistance of the furnace is

$$R = \frac{V}{I} = \frac{120}{5} = 24 \text{ ohms.}$$

The amount of heat evolved during 5 hours, or $5 \times 3600 = 18,000$ sec, will then be

$$H = 0.24I^2 \, Rt = 0.24 \times (5)^2 \times 24 \times 18,000 = 2.592,000$$

calories or 2.592 kilo-calories.

Apart from the current, the heating of conductors depends also on their cross-sectional area. It is often said that the heating of conductors depends on the density of the current, i.e., the amperes per square mm of cross-sectional area. Thus, if a current $I = 100$ A flows through a conductor with a cross-sectional area $s = 10$ mm², the current density is

$$i = \frac{I}{s} = \frac{100}{10} = 10 \text{ A/mm}^2.$$

The amount of heat developed in a conductor increases with increasing current density. Conductors must therefore be selected on the basis of the maximum permissible current density at which their temperature will not rise to a value dangerous for the insulation.

For one reason or another the insulation of wires or cables may be damaged. In such a case, the damaged spot will offer least resistance to the flow of current, and the current will find a shortened return path and bypass the load (Fig. 5). Because of this the resistance of the circuit will sharply drop, and the current will rise many times. This rise in current may damage the insulation of the conductors

Fig. 5. Short in an electric circuit

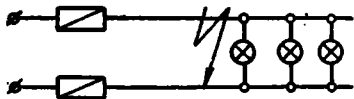remaining in circuit or even melt the conductors. This condition is called a short circuit. Whenever a short circuit occurs, the faulted circuit must be immediately disconnected from the supply if grave damage is to be avoided. This purpose is served by fuses or circuit breakers connected in series with the circuits. A fuse consists of a fusible element, or link, of such diameter that the link will melt before the temperature of the conductor becomes dangerous for the insulation. In melting (or "blowing"), the fuse automatically disconnects the faulted circuit from the supply. Circuit breakers, for the same purpose, are fitted with overcurrent devices.

3-8. Power and Energy of an Electric Current

Power—the rate of doing work—is equal to voltage multiplied by current. This applies to direct current circuits, the case for a.c. circuits will be explained later in this chapter.

The unit of electrical power is the watt (W) or the kilowatt (kW); 1 kW = 1000 W. Mechanical power is expressed in horsepower (hp), especially that of various engines. One horsepower is equal to 0.736 kW. Power measurements are taken with a wattmeter or a kilowatt-meter.

What we pay for, however, is consumed energy, not power. Energy is the ability to do work, and so we pay for the work that electricity does for us. The unit of electrical energy (or work) is the watthour (Wh) or the kilowatthour (kWh); 1 kWh = 1000 Wh. Electric energy consumption is measured with watthour meters, or simply meters. Consumed or delivered energy can be found by multiplying power by time.

The power expended in a direct current circuit in watts, is equal to the product of the volts applied to the circuit by the amperes flowing through it, one watt being equal to one ampere multiplied by one volt.

Example 8. Find the cost of the electric energy taken by an electric motor with a power rating $P = 10$ kW during 8 hours. The rate of payment is 1.1 kopeks per kWh.
The energy consumption is
$$W = Pt = 10 \times 8 = 80 \text{ kWh.}$$
The cost of the energy then is
$$80 \times 1.1 = 88 \text{ kopeks.}$$

3-9. Efficiency

Every kind of machine always consumes or loses part of the energy which it receives for its operation. When these losses are small, we say the machine has a high efficiency.

Efficiency is defined as the ratio of the power output (or useful power) P_{out} to the power input (or available power) P_{in}. Since no machine is able to deliver more energy than it receives, the efficiency is always less than unity.

In electric motors losses occur because of the heating of the windings and cores, friction in the bearings, and other causes. The less such losses, the greater the efficiency.

Efficiency is denoted by the Greek letter η (eta).
$$\eta = \frac{P_{out}}{P_{in}} .$$

Example 9. The power input to an electric motor is $P_{in} = 13.4$ kW, while its power output is $P_{out} = 11.4$ kW.
Determine the efficiency and the losses of the motor.
The motor efficiency is
$$\eta = \frac{11.4}{13.4} = 0.85.$$
The losses in the motor are
$$13.4 - 11.4 = 2 \text{ kW.}$$

Example 10. Determine the cost of the energy consumed by an 18-kW motor if it operates continuously for 8 hours, its efficiency $\eta = 0.87$ and the rate of payment is 1.1 kopeks per kWh.
The output power of the motor (P_{out}) is its rating, or 18 kW.

The required input power then is

$$P_{in} = \frac{18}{0.87} = 20.7 \text{ kW}.$$

Energy consumption during 8 hours will be

$$20.7 \times 8 = 165.6 \text{ kWh}.$$

The energy bill will therefore be

$$165.6 \times 1.1 \approx 182.2 \text{ kopeks}.$$

3-10. Electromagnetic Induction

Magnetism. Since ancient times it has been known that certain kinds of iron ore possess the ability to attract small pieces of iron or steel; this ability is called magnetism and pieces of ore exhibiting this property are called natural magnets or lodestones. It is also possible to make an artificial magnet by stroking a steel bar with a magnet several times in the same direction. The steel bar becomes magnetized. If soft steel filings are sprinkled on a bar or plate of magnetized steel, it will attract and hold the filings. The attraction for the filings will be greatest at two ends as illustrated in Fig. 6. The two ends that have the greatest attraction for the iron filings are called the poles of the magnet. If we magnetize a steel needle and mount it on a pivot

Fig. 6. Magnetized steel bar attracts iron filings

at its centre, the needle will turn until one pole points to the north and the other to the south. By convention, the north-seeking end is termed the North pole (N), and the south-seeking end,
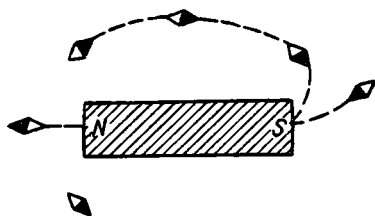
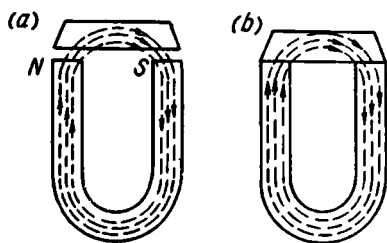Fig. 7. Compass needles in the field of a magnet

Fig. 8. Horse-shoe magnet:
a—magnetic circuit open; b—magnetic circuit closed

the South pole (S). If we bring the north-seeking end of a compass needle near to the north pole of a magnet, the needle will be repulsed, while it will be attracted by the South pole of the magnet. Thus, like poles repel and unlike poles attract.

If several tiny compass needles are placed at various points near a magnet, each needle will take up a different position (Fig. 7), thereby demonstrating that there are forces acting upon the needles in the space surrounding the magnet and that these forces act along definite lines of direction. This property of the space around a magnet is called 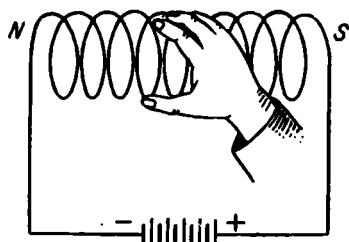a magnetic field and the lines along which the forces act are called lines of magnetic force. By convention, it is assumed that lines of magnetic force leave a magnet at the North pole and enter it at the South pole. All the lines of magnetic force leaving any given pole or passing through a given section constitute the magnetic flux.

Fig. 9. Using the right-hand rule for determining the poles of a solenoid

If a steel plate is brought near the poles of a horse-shoe magnet, nearly all the magnetic flux will pass through the plate (Fig. 8*a*) because a metal offers less opposition to magnetic flux than does air. If the plate is placed on the poles a closed magnetic circuit will be produced (Fig. 8*b*) which will display no magnetic properties because it has no poles.

Electromagnetism. The current flowing in a conductor sets up a magnetic field around the conductor. If the current-carrying conductor is wound into a helical coil, the coil acquires the properties of a
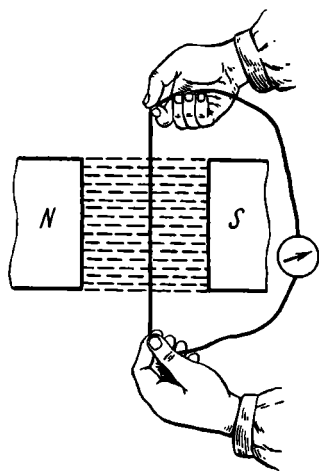
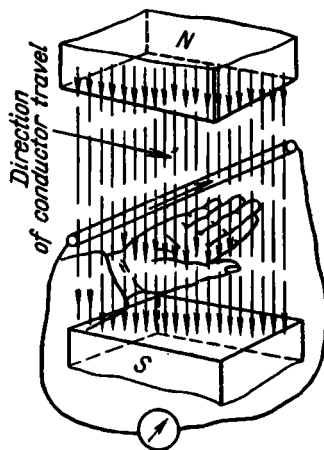*Fig. 10.* Induction of an emf in a conductor

Fig. 11. Using the right-hand rule to find the direction of an induced current

magnet and is called a solenoid. Like a magnet, it will have two poles, north and south. The poles of a solenoid can be identified by the so-called right-hand rule: place the right hand on the solenoid so that the four fingers point in the direction of current flow through the coil; the spread thumb will point in the direction of the north pole (Fig. 9).

A steel core placed inside a solenoid becomes magnetized by the magnetic field of the solenoid. A solenoid containing a steel core is called an electromagnet.

When a conductor is moved through a magnetic field so that it cuts the lines of magnetic force (Fig. 10), an emf is induced in the conductor. This phenomenon is known as magnetic induction. When the circuit connected to the conductor is closed, the indicating instrument in the circuit will indicate a current flowing around the circuit. An emf will also be induced if the conductor remains stationary and the magnetic field moves relative to it. The direction of the induced emf or current can be found by the right-hand rule (Fig. 11): place the right hand so that the lines of magnetic force enter the palm, extend the thumb in the direction of the motion of the conductor; then the four outspread fingers extended will give the direction of the induced emf (current).

3-11. Principle of Operation of a D. C. Generator and a D. C. Motor

A generator is a machine which converts the mechanical power of a prime-mover (turbine, steam engine, internal combustion engine, etc.) into electric power. A d. c. generator consists of a field structure which sets up the magnetic field, and an armature carrying the main winding and a device called the commutator.

The field structure consists of a frame (or yoke) to which the pole cores carrying the coils of the field winding are attached. The armature has a core assembled of electrical steel laminations. The longitudinal slots on the periphery of the core receive the armature conductors. When the armature is rotated through the magnetic field created by the field winding, its conductors cut the lines of magnetic force and an emf is induced in each of them. Since each armature conductor alternately passes poles of different polarity, the current induced in it alternately reverses its direction. This means that an alternating current is induced in the armature winding. To rectify the alternating current to direct current, d.c. generators incorporate a commutator. The principle of operation of the commutator is illustrated in Fig. 12. Referring to the figure, conductors *ab* and *cd* connected in series to form a loop or turn rotate in the magnetic field. The ends of the conductors are connected to two half-rings *A* and *B*,

separated and insulated from each other. Two stationary brushes, *1* and *2*, ride on the two half-rings. When conductor *ab* passes under the North pole, the induced current flows as indicated by the arrows. At this moment the current is led out to the external circuit by brush *1* and enters the loop at brush *2*. Thus brush *1* is positive and brush *2* negative. As the turn goes through another 180 degrees, conductor *ab* comes under the South pole, and a current opposite in direction
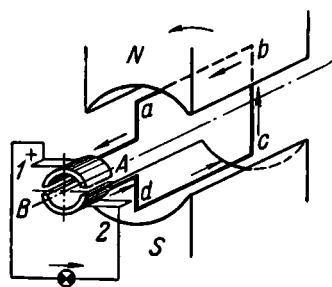
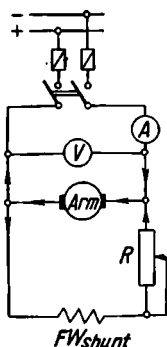Fig. 12. The principle of opera-
tion of the commutator

Fig. 13. Schematic
circuit diagram of
a d.c. shunt-wound
generator
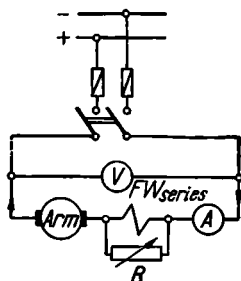
Fig. 14. Schematic
circuit diagram of a
d.c. series-wound gene-
rator

will now be induced in it. At this moment, however, half-ring *B* comes in contact with brush *1* and half-ring *A* with brush *2* and the connections of the conductors to the external circuit are reversed. As a result, the current in the external circuit will remain unchanged in direction, i.e., the load will carry a continuous current flowing in one and the same direction, although an alternating current is induced in the rotating loop or turn.

Commercial generators have a large number of turns placed in the magnetic field instead of one loop. Correspondingly, the commutator has a large number of bars insulated from one another by segments of mica or micanite.

As to the method of connecting the field winding, i.e., the winding which produces the magnetic field, to the armature winding, d.c. generators are classed into three types: shunt-wound, series-wound, and compound-wound.

1. *Shunt-wound generators* (Fig. 13). In these machines the field winding FW_{shunt} consists of coils wound with a great number of turns of fine wire and is connected in parallel, or shunt, with the armature *Arm*. The field winding is connected in series with a field rheostat *R* to adjust the generator voltage.

2. *Series-wound generators* (Fig. 14). In these machines the field winding FW_{series} consists of coils wound with a small number of turns of heavy wire and is connected in series with the armature *Arm.*
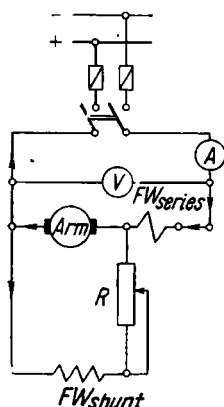
Fig. 15. Schematic circuit diagram of a d.c. compound-wound generator
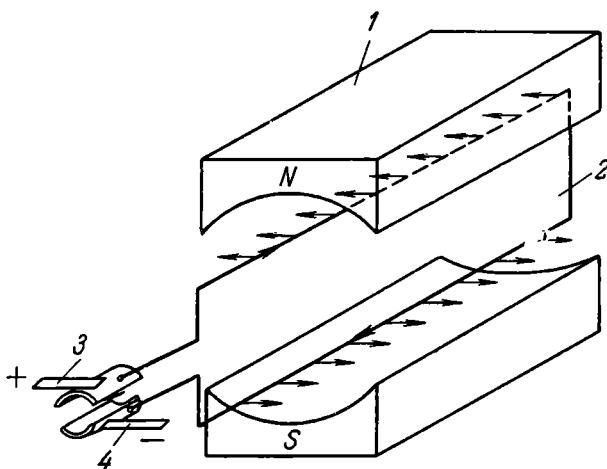
Fig. 16. Principle of operation of a d.c. motor

The field rheostat R is connected in parallel with the field winding.

3. *Compound-wound generators* (Fig. 15). In these machines each pole carries two windings, one, FW_{shunt}, consisting of a great number of turns of fine wire, the other FW_{series}, of a few turns of heavy wire. The first winding is connected in parallel and the second in series with the armature.

If a d.c. generator receives direct current from some external source, its armature will rotate, i.e., the generator will become a motor. An electric machine capable of operating as a generator and a motor is termed a reversible electric machine.

The principle upon which an electric motor operates is diagrammatically represented in Fig. 16.

A coil 2, free to rotate about a horizontal axis, is arranged in the magnetic field of field structure 1. The ends of the coil 2 are connected to two half-rings on which brushes 3 and 4 ride to feed the coil with current from a supply circuit.

When the coil is energized with a direct current, its top and bottom sides are acted upon by electromagnetic forces which tend to push out the sides from under the poles. Since the coil is pivoted, these forces can only turn the coil through some angle. The direction of the

electromagnetic forces can be determined by the left-hand rule: place the left hand so that the lines of magnetic flux enter the palm, extend the four fingers in the direction of the current in the conductor; then the extended thumb will show the direction of the forces. In our example, the force acting on the upper side of the coil is directed to the left and that acting on the bottom side is directed to the right. The frame must therefore turn counterclockwise.

As the frame turns, its sides cut the lines of magnetic flux and an emf is developed in the frame.

Let several coils, similar to the one above, be secured on a shaft so that they are uniformly spaced between the poles and the ends of each coil are connected to a pair of plates (in place of half-rings) on which two stationary brushes ride. Then, at any given instant of time, only one coil will receive supply current through the brushes and its pair of contact plates. As it turns and passes through a certain angle, the brushes come in contact with the next pair of contact plates, and the next coil is energized. This sequence is repeated as each successive coil replaces its predecessor, and continuous rotation is maintained.

In a d.c. electric motor, instead of a series of coils, there is a slotted steel core (armature) which carries a large number of conductors connected with each other and also to the bars of a commutator in a certain manner.

By the right-hand rule, the emf induced in the upper side of the coil (Fig. 16) is directed toward the reader, that is, opposes the current flowing into the conductor. Such an emf is known as a counter-emf.

In the conductor, therefore, both the applied voltage V and an induced emf E are present simultaneously. By Ohm's Law, the current flowing in the conductor will be the difference in applied voltage and counter-emf divided by the resistance R_{arm} of the armature winding, or

$$I = \frac{V - E}{R_{arm}}.$$

Consequently, the difference between the applied voltage and the counter-emf is equal to the voltage drop across the armature winding:

$$V - E = I R_{arm}.$$

Since the voltage drop across the armature winding is negligible, the counter-emf differs only slightly from the applied voltage.

As the motor just starts up, the counter-emf is nil, and the current is determined only by the applied voltage and the armature winding resistance. If no measures were taken, the current at starting could become large enough to damage the winding. For example, a 10-kW

motor with a voltage rating of 110 V has an armature resistance of about 0.1 ohm. The starting current of such a motor could be

$$I = \frac{V}{R_{arm}} = \frac{110}{0.1} = 1100 \text{ A},$$

whereas the normal running current of the motor is only about 100 A. To limit the starting current, every motor has a starting rheostat connected in series with the armature circuit.

Like d.c. generators, d.c. motors may have their fields connected in shunt, series, or compound, and are known, respectively, as shunt-wound, series-wound, and compound-wound motors.

B. ALTERNATING CURRENT

3-12. General

When a coil is rotated in a magnetic field, an alternating emf is induced in it. If the coil is closed across a load, an alternating current will flow around the circuit (Fig. 17).

1 et us see what occurs as the coil turns within a magnetic field. As the coil passes under the North pole (position *I*), the greatest value of emf is induced because it cuts the maximum number of lines of magnetic flux. The resultant current flows from terminal *A* to ter-

Fig. 17. Principle of operation of an a.c. generator
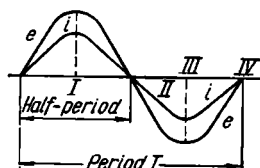
Fig. 18. Periodic variations of current and emf

minal *B* through the external circuit. When the coil reaches position *II*, it moves along the lines of magnetic flux without cutting them. The emf in the coil and, hence, the current in the circuit are therefore nil. When the coil passes under the South pole (position *III*), the emf again reaches a maximum, but in the opposite direction. Now, the current in the external circuit flows from terminal *B* to terminal *A*. At position *IV* the emf again drops to zero. This cycle of events is repeated during each full revolution and, consequently, a current is produced which changes in magnitude continuously and reverses in

direction at regular intervals. Such a current is called an alternating current.

Fig. 18 gives a graph of changes in current i and emf e during one revolution of the coil. In position *I* the emf and current reach their maximum positive values (the curves at this point being above the X-axis). At this moment the coil passes directly under the North pole. In position *II* the emf and current drop to zero. As the coil rotates farther on, the sign of the emf and current changes because the coil comes under the South pole (the curves of the emf and current now lie below the X-axis). When position *IV* is reached, the emf and the current again drop to zero.

The time during which the emf or the current goes through a full cycle of changes in direction and magnitude, is called the period and is designated by the letter T (Fig. 18). The number of periods per second is called the frequency and denoted by the letter f. Frequency is stated as so many cycles per second (c.s). The frequency and the period are reciprocals, i.e.,

$$T = \frac{1}{f}; \qquad f = \frac{1}{T}.$$

Thus the duration of one cycle of alternating current with a frequency of 50 cycles per second is

$$T = \frac{1}{f} = \frac{1}{50} = 0.02 \text{ sec}.$$

3-13. Self-induction. Resistance and Reactance of A. C. Circuits

An alternating current produces an alternating magnetic field. Both when building up and collapsing, the magnetic lines of force of this field will cut across the conductor to induce an emf in it. This emf is called the emf of self-induction. The direction of the emf of self-induction is found by Lenz's rule. This rule states that the direction of an induced emf or current is always such that it opposes the cause due to which it appears.

During the first quarter of a cycle (Fig. 19) the current rises, and the induced emf, according to Lenz's Law, opposes the applied voltage, acting against the current rise. In other words, the emf of self-induction is negative. At point *2* the rise in current ceases and the emf of self-induction drops to zero. During the second quarter of a cycle the current continuously decreases and the emf of self-induction reverses, now tending to prevent the current from decreasing. The emf of self-induction will thus be positive throughout the second quarter of a cycle and reach a maximum at point *3*. During the third quarter of a cycle the current reverses to reach a negative maximum.

The emf of self-induction still remains positive to oppose the further current change in the negative direction, but gradually decreases to zero at point *4* where the current ceases its negative increase. Past point *4* the emf becomes negative because the current is now rising to a positive maximum.
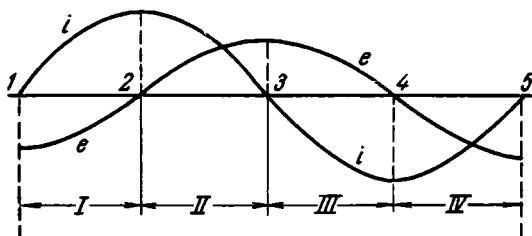

Fig. 19. Periodic variations of current and induced emf:
I, II, III, IV—cycle quarters

From the above, we can see that "the curves, or waveforms, of current and emf of self-induction are out of phase", i.e., they reach their corresponding zero, maximum and intermediate values at different instants. Since electrical degrees are proportional to time, it is said that the two quantities are out-of-phase, have a phase difference of some degrees. Thus, the emf of self-induction lags behind the current by one quarter of a period in time, or 90 degrees.

If an unvarying current is passed through the coil, no self-induction will be observed, because the magnetic field of the coil remains unvarying, too. In this case the applied voltage is expended only to overcome the coil resistance. This is the voltage drop across the d.c. (ohmic) resistance of the coil, or the resistive voltage drop. However, if an alternating current flows through the same coil, an emf of self-induction opposing the applied voltage is developed. Therefore, a higher voltage will be required to send an alternating current of the same magnitude as the previous direct current through the coil so that it is high enough to overcome both the resistance of the coil and the opposition due to the emf of self-induction.

It may therefore be said that in an a.c. circuit, to the ohmic resistance of the coil which depends on the material and size of the conductor, is added the opposition due to the emf of self-induction. The latter is termed inductive reactance and designated by the letter X_L. Practical a.c. circuits may also contain capacitance. The effect of capacitance in an a.c. circuit is that it produces additional opposition to the flow of the current. The opposition to alternating current

produced by capacitance is called capacitive reactance and is designated by the letter X_C. The combined effect of resistance R, inductive reactance X_L, and capacitive reactance X_C is termed impedance.

For a circuit containing resistance, inductance and capacitance

$$Z = \sqrt{R^2 + (X_L - X_C)^2} \text{ (ohms)}.$$

For a circuit containing resistance and inductance

$$Z = \sqrt{R^2 + X_L^2} \text{ (ohms)}.$$

For a circuit containing resistance and capacitance

$$Z = \sqrt{R^2 + X_C^2} \text{ (ohms)}.$$

Ohm's Law for an alternating current circuit can be stated in the following way: the value of an alternating current I is equal to the applied voltage V divided by the impedance Z of the circuit, i.e.,

$$I = \frac{V}{Z}.$$

In an a.c. circuit containing resistance and inductance the current lags behind the voltage by a certain angle, while in an a.c. circuit containing resistance and capacitance the current leads the voltage. This angle of lag or lead is often designated by the Greek letter φ (phi) and is termed the phase displacement or phase angle.

3-14. Power in A. C. Circuits

If the current in, and the voltage across an a.c. circuit were in phase, the power developed by an alternating current could be determined by the formula for direct currents, i.e., by multiplying the voltage by the current. Since the phase angle does exist, a correction factor has to be introduced having the mathematical form of $\cos \varphi$ (the cosine of phi). The formula of a.c. power then becomes

$$P = VI \cos \varphi \text{ (watts)}.$$

Cos φ mathematically takes account of the decrease in power in comparison with a case when there is no phase displacement between the voltage and current. It is called the power factor and is always less than unity. An understanding of the reason why this is so can be gained from reference to Fig. 20. Fig. 20a shows the current and voltage of a purely resistive circuit only containing several electric lamps. There is no phase angle between the voltage and current in this case. Since the power is equal to the product of the current and the voltage, and the product of two negative quantities (during the second half-cycle) is positive, the power will always be positive.

Fig. 20*b* gives the waveforms of current and voltage in a circuit containing inductive reactance which produces a phase displacement of φ degrees. Since multiplication of a positive quantity by a negative quantity results in a negative product, the power becomes negative at certain intervals of time. This means that there are moments when the load returns some power to the line at the expense of the energy
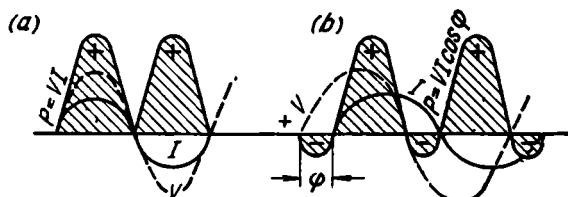

Fig. 20. Current and voltage waveforms in two
circuits:
a—circuit without inductive reactance; *b*—circuit containing inductive reactance

stored up in its magnetic fields during the intervals of rise in current (shaded areas marked with a minus sign). Thus, out of the total power, $S = VI$, received by the load, only $P = VI\cos\varphi$ is used to perform useful work. The remaining portion of the total, or apparent, power goes to produce magnetic fields during rise in current and is returned when the current falls. The former is called active or true power, and the latter is known as the reactive power.

The apparent power input to an a.c. circuit is determined by the formula

$$S = \sqrt{P^2 + Q^2},$$

where S = apparent power in VA or kVA (volt-amperes or kilovolt-amperes);

P = active or true power in W or kW (watts or kilowatts);

Q = reactive power in VAr or kVAr (volt-amperes reactive or kilovolt-amperes reactive).

3-15. Three-phase Currents. Star and Delta Connections. Power of Three-phase Currents

A three-phase current consists of three single-phase alternating currents that differ in phase by 120 degrees. To obtain a three-phase current, it is necessary to arrange three windings (*I*, *II* and *III*)

spaced 120 degrees apart round the armature (stator) of a generator (Fig. 21). These windings may be either star- or delta-connected.

For star connection (Fig. 22) all the ends, or finishes, of the phase windings are connected together to form a neutral point, while the beginnings, or starts, of the phase windings are left as terminals for the line wire (i.e., conductors running from the generator to the load).
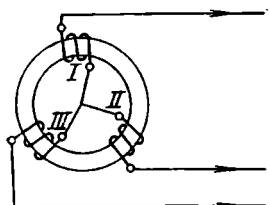
Fig. 21. Three windings on the armature (stator) of a generator
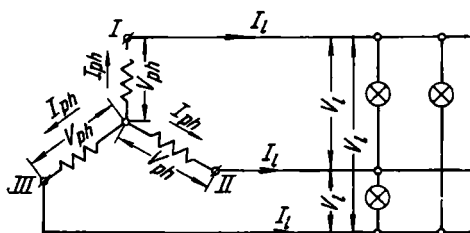
Fig. 22. Star connection

For delta connection (Fig. 23) the finish, F_1, of the first phase is connected to the start, S_2, of the second phase; the finish F_2 of the second phase to the start S_3 of the third phase; and the finish F_3 of the third phase to the start S_1 of the first phase. The phase junctions form the terminals for the line wires.
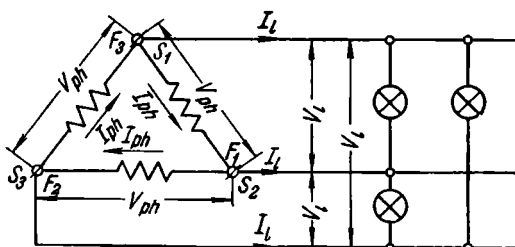
Fig. 23. Delta connection

In a star-connected circuit, the current I_{ph} in each phase (the phase current) is equal to the current I_l flowing in each of the three line wires connecting the generator with its load (the line current). The voltage of each phase V_{ph} (the phase voltage), however, will be 1.73 times less than the voltage V_l (the line voltage) measured between any two of the line conductors.

Therefore, in a star-connected circuit the phase and line voltages and currents are related thus:

$$I_{ph} = I_l; \quad V_{ph} = \frac{V_l}{1.73}.$$

In a delta-connected circuit the line current I_l is 1.73 times the phase current I_{ph}, while the phase voltage V_{ph} remains equal to the line voltage V_l. The relations between the phase and line currents and voltages in a delta-connected circuit then are:

$$I_{ph} = \frac{I_l}{1.73} \quad \text{and} \quad V_{ph} = V_l.$$

Three-phase electric motors are usually wound so that their phases can be connected either into a star or a delta.

For example, a conveyor motor, when star-connected, can operate on a supply voltage of 660 V. For operation on 380 V its windings are delta-connected.

With star connection:

$$V_{ph} = \frac{V_l}{1.73} = 380 \text{ V}.$$

With delta connection:

$$V_{ph} = V_l = 380 \text{ V}.$$

In both cases, each phase receives the same voltage of 380 V.

The active or true power of a three-phase system is given by the formula

$$P = 1.73 \, VI \cos \varphi \text{ (watts)},$$

where V and I are the line current and voltage, respectively.

The use of three-phase current requires installation of three line wires, whereas a single-phase circuit only requires two wires. Yet three-phase current has many advantages to offer. The most valuable property of three-phase current is its ability to produce a revolving magnetic field—the basis of induction-motor operation. This property was discovered by M. O. Dolivo-Dobrovolsky of Russia who built the first induction motor, which has now become the main type of motor used in all industries.

In mines, three-phase electric power serves to feed the motors of nearly all motor-driven mechanisms and machines. The exceptions to this are the electric locomotives and certain types of hoisting and winding installations.

Example 11. The drive motor of a scraper-chain conveyor operates with a load of 19 kW at 380V. Find the current drawn by the motor if its efficiency is 0.93 and its power factor (cos φ) is 0.85.

Power input to the motor:

$$P = \frac{19}{0.93} = 20.4 \text{ kW, or } 20,400 \text{ W}.$$

The load current then is

$$I = \frac{P}{1.73 V \cos \varphi} = \frac{20,400}{1.73 \times 380 \times 0.85} = 36.6 \text{ A}.$$

Example 12. The electric motor of a booster ventilating fan is connected to 380-volt supply. During its operation an ammeter indicates that it draws a load current of 10 A. Find the power input to the motor if its power factor (cos φ) is 0.8

$$P = 1.73\,VI\cos\varphi = 1.73 \times 380 \times 10 \times 0.8 = 5280 \text{ W or } 5.28 \text{ kW.}$$

3-16. Transformation of Alternating Current

From the formula for the power of three-phase alternating current $P = 1.73\,VI\cos\varphi$ it can be seen that a given power can be delivered at various values of voltage, and that the higher the voltage selected
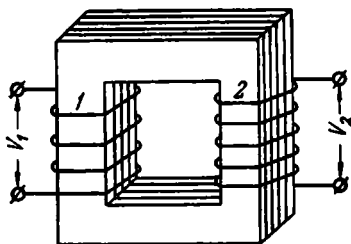
Fig. 24. Connection diagram of a single-phase transformer
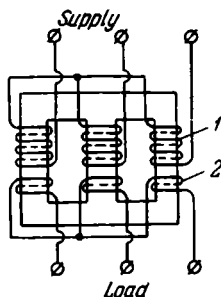
Fig. 25. Connection diagram of a three-phase transformer

the less the current and the less the size of the conductors to carry it. A decrease in the cross-sectional area of the conductors leads to a decrease in the weight of the conductor copper and, hence, to lower costs of erection of overhead transmission lines or underground cables. It is, therefore, advantageous to transmit electric power at high voltages. The magnitude of voltage is selected in accordance with the length of the transmission line and the amount of power to be transmitted. For large distances and large blocks of power, the transmission line voltage will be of the order of 35, 110 or even 500 kV. For example, the power transmission lines from the Kuibyshev hydroelectric station on the Volga operate at 500 kV. However, a.c. generators are designed for voltages of 6.6 to 10.5 kV. Before the power can be transmitted, the generator voltage has to be raised, or stepped up. This is done by apparatus known as transformers. Transformers are also necessary at the receiving end, but this time to step down the voltage to a value convenient for supply of motors and lighting equipment.

A transformer consists of a laminated steel core (Fig. 24) and two windings: a primary *1* which is connected to the source of power and

a secondary *2* which delivers power to the load. The alternating current, in flowing through the primary winding, produces an alternating magnetic flux in the core and links the secondary winding.

Since the magnetic flux alternates continuously, an alternating emf E_2 is induced in the secondary. This flux simultaneously links the turns of the primary and therefore induces in it a counter-emf E_1 which opposes the applied voltage V_1. The current flowing in the primary is hence determined by the difference between the supply voltage V_1 and the counter-emf E_1.

The number of turns in each winding bears a definite relation to the emfs induced in them:

$$\frac{E_1}{E_2} = \frac{n_1}{n_2}.$$

Thus, the greater the number of turns n_1 in the primary, the greater emf E_1; or the less the number of turns n_2 in the secondary, the smaller emf E_2.

A voltmeter connected across the primary or the secondary will read the terminal voltage V_1 or V_2 but not the emf's. The difference, however, will be very small, and the terminal voltages of the primary and secondary at no-load (i.e., with the secondary open) bear the same ratio as the turns in the respective windings. This ratio is known as the transformation, or turns, ratio

$$k = \frac{V_1}{V_2} = \frac{n_1}{n_2}.$$

In three-phase circuits the voltage is stepped up or stepped down with three-phase transformers. A three-phase transformer (Fig. 25) consists of a core with three legs or limbs on which the primary windings *1* and secondary windings *2* of each phase are placed. The windings may be either star- or delta-connected. The entire assembly consisting of the core and the windings is placed in a tank filled with transformer oil which serves as a coolant. The transformer oil simultaneously reliably insulates the turns of the windings from each other and the windings, as a whole, from the core iron. To increase the rate of cooling, transformer tanks are constructed with finned radiating surfaces or radiators.

The active output power of a three-phase transformer is

$$P_2 = 1.73 V_2 I_2 \cos \varphi_2,$$

where V_2 and I_2 = line voltage and current of the secondary circuit;

       φ_2 = phase angle between current and voltage in the secondary circuit.

The active input power to the transformer is

$$P_1 = 1.73 V_1 I_1 \cos \varphi_1,$$

where V_1 and I_1 = line voltage and current of the primary circuit;
    φ_1 = phase angle between current and voltage in the
        primary circuit.

Input power P_1 and output power P_2 are measured in watts or kilowatts.

The power factor $\cos \varphi_2$ in the formula of the transformer output power depends upon the nature of the load carried by the transformer. For a load consisting of lighting equipment or electrical heating appliances, $\cos \varphi_2 = 1$. If electric motors are connected to the secondary circuit, $\cos \varphi_2$ is always less than unity. Therefore, the power of a transformer is best expressed in a form independent of the power factor of the secondary circuit, i.e., by the formula

$$S_2 = 1.73 V_2 I_2.$$

As the apparent power S_2 is the product of current in amperes and voltage in volts, the unit of apparent power for transformer rating is the volt-ampere (VA) or the kilovolt-ampere (kVA); 1 kVA = 1000VA.

If the power rating of a transformer is 100 kVA, it means that the transformer can supply 100 kW of power to lighting or heating appliance loads. But, if this same transformer is used to supply electric motors at a power factor ($\cos \varphi_2$) of 0.75, the load on it must not exceed 75 kW ($P_2 = 100 \times 0.75 = 75$ kW), to avoid dangerous heating of the transformer.

3-17. *Autotransformers*

An autotransformer is a transformer which has only one winding. It is connected across the high supply voltage (points a and b in Fig. 26), and the low output voltage is taken from taps connected to a certain portion of the same winding (points a and c).

Autotransformers have certain advantages in that they require less copper in the winding and less insulating material because there is no insulation between the high- and low-voltage portions. Their main disadvantage is that the winding directly connects the high- and the low-voltage sides.
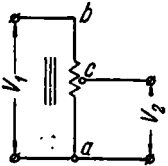
Fig. 26. Circuit diagram of an autotransformer

In mines autotransformers find application as a means for starting squirrel-cage induction motors at a reduced voltage.

3-18. A. C. Electric Motors

The Induction Motor. For its operation an induction (or asyn-chronous) motor depends on a revolving magnetic field.

The revolving magnetic field is produced by supplying the three-phase windings of the motor with three-phase current. This field re-volves at what is known as synchronous speed, i.e., in synchronism with the power supply frequency. More specifically, the synchronous speed, n_1, of the magnetic field is a function of the supply frequency, f, and the number of pole pairs, p (one north and one south pole con-stitute a pole pair):

$$n_1 = \frac{60\,f}{p} \text{ (rpm)}.$$

Thus, the synchronous speed of the magnetic field of a four-pole motor when supplied with 50-c/s current, will be

$$n_1 = \frac{60 \times 50}{2} = 1500 \text{ rpm}.$$

An induction motor consists of a stationary frame with a stator core, which carries windings energized with three-phase alternating current, and a rotor. The stator windings may be either star- or delta-connected. The rotor, in general, is a cylinder with longitudi-nal slots in its periphery to receive the rotor winding. The latter also may be star- or delta-connected. When the magnetic field of the stator revolves, its lines of force cut the rotor winding to induce emfs in it. Since the rotor winding is connected into a closed circuit, the induced emf's cause current to flow through it and thereby pro-duce a magnetic field around the rotor.

The revolving field set up by the stator interacts with the field of the rotor to produce a torque which makes the rotor follow it at a slightly lower (asynchronous) speed.

The speed at which the revolving field cuts the rotor conductors is equal to the difference between the synchronous speed, n_1, and the actual rotor speed, n_2. The ratio of this difference to the field, or synchronous, speed is termed the slip and is denoted by the letter s.

Thus, the slip is

$$s = \frac{n_1 - n_2}{n_1}.$$

The slip is usually expressed in per cent. Then

$$s^0/_0 = \frac{n_1 - n_2}{n_1} \times 100.$$

Example 13. Find the slip of a 50-cycle induction motor, if its winding has $p = 3$ pole pairs and the rotor runs at a speed $n_2 = 985$ rpm.

The speed of the revolving magnetic field is

$$n_1 = \frac{60 \times f}{p} = \frac{60 \times 50}{3} = 1000 \text{ rpm}.$$

The slip will be

$$s = \frac{n_1 - n_2}{n_1} \times 100 = \frac{1000 - 985}{1000} \times 100 = 1.5\%$$

Example 14. Determine the speed of the electric motor driving a coal-cutting machine if it is known that the stator winding has four poles and the slip is 2.3 per cent.

The speed of the revolving field is

$$n_1 = \frac{60 \times f}{p} = \frac{60 \times 50}{2} = 1500 \text{ rpm} \quad (f = 50 \text{ c/s}).$$

To find the speed n_2 of the rotor, we use the slip formula

$$s = \frac{n_1 - n_2}{n_1} \times 100,$$

from which

$$n_2 = \frac{100 - s}{100} n_1 = \frac{100 - 2.3}{100} \times 1500 = 1465 \text{ rpm}.$$

In accordance with rotor construction, induction motors are of two types: squirrel-cage and slip-ring.

The Squirrel-cage Induction Motor. In a squirrel-cage motor, the rotor winding consists simply of conducting bars embedded in slots in the iron core of the rotor and connected together at each end by means of short-circuiting rings. The rotor winding thus forms a complete closed circuit in itself. This structure resembles a squirrel cage—a fact to which this type of motor owes its name.

The bars and short-circuiting rings of the squirrel-cage structure may be made of copper, but recent practice favours direct casting of aluminium in the slots of the rotor core. The rotor core is built up of separate thin sheet-steel laminations insulated from each other by paper or varnish in order to reduce eddy-currents which retard rotation of the rotor. For better cooling, the core is made with ventilating ducts. The majority of coal mining machines are equipped with squirrel-cage induction motors.

Slip-ring Induction Motor. The rotor of a slip-ring induction motor has three phase windings (similar to those of the stator). Leads from the rotor winding are brought out to three slip rings seated on the shaft so that they are insulated both from the shaft and from each other. The slip rings carry a set of brushes which are connected to a rheostat. The latter is used to increase the resistance of the rotor winding and thereby limit the starting current.

As a slip-ring induction motor comes up to speed, the starting current decreases, and the rheostat is gradually cut out, and the rotor winding is short-circuited with the device provided for this purpose at the slip rings.

Fig. 27 shows the main elements of this device and its operation. It consists of a shorting ring *1* with three contacts *2*. Rotation of the

handle *4* on the shaft *5*, shifts the ring *1* along the rotor shaft *3* until it short-circuits the rings *6* with its contacts.

So as to avoid unnecessary wear of the brushes, a simultaneous function of the short-circuiting device is to lift the brushes from the slip rings after it has short-circuited them.

Slip-ring induction motors are used in coal mines to drive hoisting winches, winding machines, compressors, main ventilating fans, etc.

Load of an Induction Motor. As the load of an induction motor increases, the speed of its rotor decreases (its slip becomes greater). As a result, the revolving magnetic field cuts the rotor winding at a higher speed and a greater rotor emf is induced, accompanied by an increase in the rotor current. This increase then leads to a rise in the stator current. Minimum current is taken by the motor when it operates at no-load. When the rotor is locked, the motor draws maximum current.

Fig. 27. Short-circuiting of slip rings: *a*—slip rings unshorted; *b*—slip rings shorted

The Starting of an Induction Motor. A squirrel-cage induction motor is generally started on full voltage (direct-on-line starting). At the very first instant the rotor is at a standstill (the slip $s = 1$) and the revolving magnetic field cuts the rotor winding at full speed, inducing in it a large emf. Since the rotor winding is short-circuited, a very large current appears in it. The current in the stator (the starting current) is very large at this moment; it may be from 5 to 8 times rated load current. As soon as the rotor comes up to speed, the revolving magnetic field cuts its winding at a progressively lower speed, and the starting current drops. The rotors of modern squirrel-cage induction motors range in design from general-purpose motors for which large starting currents are permissible to machines designed with special squirrel cages which limit the starting current.

Reversal of Induction Motors. Since the direction in which an induction motor runs depends on the direction of the revolving magnetic field, the motor can be reversed by making the field revolve in the opposite direction. This is done by changing over the connections of any two of the supply cable leads at the stator terminals. Where the motor is controlled by a reversing magnetic starter, reversal is accomplished by pushing the "reverse" push button or switching a reversing control device. After a motor has been connected into circuit, it must be checked for proper direction of rotation. If it rotates in the wrong direction, the connections of any two of the cable leads must be changed over.
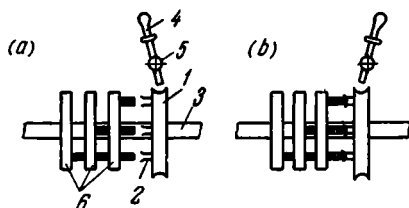
Where a motor has to be reversed frequently, this is best done by a built-in switching device (controller) incorporated in the machine operated by the motor. Controllers are used in coalcutters, and cutter-loaders. Mounted separately, they are used to operate winches.

3-19. Conversion Equipment

The conversion of alternating current to direct current in mines is effected with motor-generator sets and mercury-arc, selenium or other rectifiers.
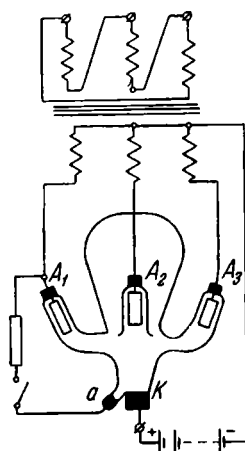
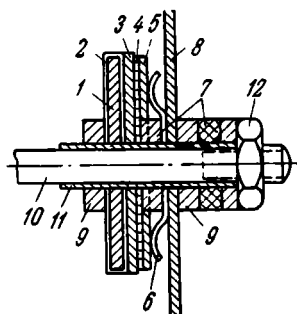*Fig. 28.* Connection of a glass-bulb mercury-arc rectifier

Fig. 29. Selenium rectifier element:

1—steel or aluminium baseplate; *2*—anti-corrosion coating; *3*—selenium layer; *4*—barrier layer; *5*—counter-electrode; *6*—spring contact washer; *7*—insulating washer; *8*—heat radiating disc (or lead); *9*—metal spacer; *10*—stud; *11*—insulating sleeve; *12*—clamp nut

A motor-generator set consists of an a.c. motor mechanically coupled to and driving a d.c. generator. The generator is connected to load circuits. Motor-generator sets are simple and reliable in service, but are big in size and relatively high in cost. Furthermore, motor-generator sets have a comparatively low efficiency due to the fact that the overall efficiency is the product of the efficiencies of each machine.

The mercury-arc rectifier (Fig. 28). A small mercury-arc rectifier consists of an evacuated glass vessel having several steel or graphite blocks A_1, A_2, A_3 (the anodes) at the top, and a pool of mercury (the cathode) at the bottom. The mercury pool is in contact with a sealed-in electrode K called the cathode. Also placed at the bottom is an ignition anode a which serves to strike an arc. An arc is drawn

by tilting the bulb several times; the mercury pool repeatedly comes in contact with the ignition anode. When the bulb comes back to normal position, an arc is formed as the mercury recedes from the ignition anode, mercury vapour is formed and the main anodes become conducting, sustaining a flow of electric current from the anodes to the cathode. The flow can only take place in one direction. Such unilateral conductivity is called valve action.

Where large currents are required, steel-tank instead of glass-bulb mercury-arc rectifiers are employed. In them, the mercury pool is contained in a steel tank.

In mines, mercury-arc rectifiers, as well as motor-generator sets, serve to supply electric locomotive overhead contact circuits and for storage-battery charging.

Selenium rectifiers consist of separate elements assembled in stacks and connected into parallel and series groups.

A selenium rectifier element (or cell) (Fig. 29) consists of a steel supporting (or back) plate one side of which is coated with a layer of selenium. After treatment with a heat and pressure process, the selenium layer is converted into two layers of different conductivity. That nearest to the steel supporting plate, or the first terminal of the element, has the higher conductivity, while the other layer, called the barrier layer, is of low conductivity. Then a tin-cadmium alloy is sprayed onto the barrier layer to form the counter electrode, or the second terminal of the rectifier. This sort of junction offers a low resistance to the flow of current when the back plate (i.e., the selenium film) is positive (which corresponds to a positive half-cycle of a.c. voltage) and a high resistance when the back plate is negative (i.e., during a negative half-cycle). This gives an intermittent pulsating current at intervals equal to one-half of the cycle. For this reason, such rectifiers are called "half-wave" and their action is referred to as half-wave rectification.

Since the barrier layer is not perfect in action, a small current is able to flow through the element during the negative half-cycle. The greater this current (known as the reverse current), the poorer the operation of a selenium element as a rectifier.

For power applications the rectifier elements are assembled into stacks. The number and connection of elements in a stack depend upon the working voltage and current rating of the rectifier. The maximum voltage permissible for one element is about 20 to 25 volts. The permissible current is a function of the diameter of the element. Elements with a diameter of 18 mm have a current rating of 0.04 A; 25, 45 and 100-mm diameter elements are rated for currents of 0.07, 0.3 and 1.5 amperes, respectively. Selenium elements can be connected into several rectifier circuits. The most widely used is the bridge connection.

Each arm of a bridge (Fig. 30) may be made up of several elements connected into series or parallel groups. In Fig. 30 each group is shown as one element *1, 2, 3* and *4*.

In circuit diagrams, it is assumed that the current flows from the triangle towards the dash line of the symbol.

The a.c. supply voltage is applied to junctions *a* and *c*. When a positive half-wave is applied to the point *a*, and a negative half-wave to the poin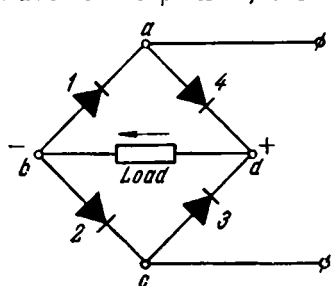t *c*, the current can only flow through element *4*, the load, and element *2* to the negative side of the supply at the point *c*. During the next half-wave, the polarity of the points *a* and *c* reverses and the current can only flow from the junction *c* through element *3*, the load and element *1* to return to the negative side of the supply at the point *a*. Thus the current flows through the load in only one direction (from *d* to *b*), irrespective of the polarity of the junctions *a* and *c*, i.e., a direct current flows through the load. Since the current flows through the load during both half-cycles, this is known as full-wave rectification and the rectifier is termed a full-wave rectifier.

Fig. 30. Single-phase bridge connection of selenium rectifier elements

Selenium rectifiers are used in mines for charging storage batteries and also in certain types of magnetic starters and relays.

For a single-phase full-wave bridge rectifier the following relations are valid:

(1) The current I_e passing through any one element is half the total current I_{rect} of the rectifier, i.e.,

$$I_e = \frac{I_{rect}}{2};$$

(2) The number of elements n connected in series in an arm of the bridge is

$$n = \frac{1.57 \, V_{rect}}{V_{rated}},$$

where V_{rect} = rectified d.c. voltage at rectifier output;
 V_{rated} = voltage per element;
(3) The a.c. voltage E_2 which must be applied to the rectifier to obtain a rectified voltage V_{rect} is

$$E_2 = 1.11 \, (V_{rect} + 1.4n).$$

Example 15. Calculate the characteristics of a selenium rectifier to be used for supply of a rectified current $I_{rect} = 5A$ at $V_{rect} = 110$ V. Assume that $V_{rated} = 20$ V.

1. The current which will flow through one element is

$$I_e = \frac{I_{rect}}{2} = \frac{5}{2} = 2.5\ A.$$

For elements with a 100-mm diameter the current rating is 1.5 A. Then the number of elements which should be connected in parallel in each arm will be:

$$a = \frac{2.5}{1.5} = 1.67\ \text{elements.}$$

We therefore take $a=2$.

2. The number of elements to be connected in series to make up one arm is:

$$n = \frac{1.57 V_{rect}}{V_{rated}} = \frac{1.57 \times 110}{20} = 8.6;$$

we therefore take $n=9$ elements

3. The total number of elements in one arm then sum up to

$$n_1 = an = 2 \times 9 = 18\ \text{elements;}$$

in all four arms

$$N = 4an = 4 \times 18 = 72\ \text{elements.}$$

The elements are assembled into four stacks of 18 elements each.

4. A. C. voltage to be applied

$$E_2 = 1.11\ (V_{rect} + 1.4n) = 1.11\ (110 + 1.4 \times 9) = 136\ \text{V.}$$

Thus, to supply this rectifier, it is necessary to use a transformer with a secondary voltage of 136 V.

The circuit arrangement of the rectifier elements may be seen in Fig. 31.

Fig. 31. Rectifier circuit of Example 15
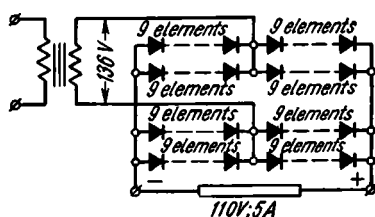
New types of semiconductor (germanium and silicon) rectifiers have of late come into wide use. A germanium or silicon rectifier is a single crystal of the semiconductor, possessing unilateral conductivity, or valve action, due to its crystalline structure. Germanium and silicon rectifiers are small in size, have high mechanical stability and excellent rectifying characteristics.

3*

Chapter IV

ELECTRICAL MEASURING INSTRUMENTS

4-1. Purposes and Types

Electrical instruments serve to measure various electrical quantities such as current, voltage, power, resistance, etc., thereby providing means to watch and check the operation of circuits and equipment.

There are also instruments designed for measuring insulation resistance. They make it possible to check on the condition of the insulation in any circuit and so detect faults in it. Special measuring bridges provide a means for determining the resistance of earth connections, and so attending personnel can readily determine whether it is low enough for adequate safety or is too high and must be lowered.

Current is measured with ammeters, voltage with voltmeters, power with wattmeters, energy consumption with watthour-meters (or simply meters), resistance with ohmmeters or bridges, insulation resistance with megohmmeters, frequencies with frequency meters, and power factor with power-factor meters or indicators.

According to the principle of operation, electrical measuring instruments are classed as: moving-iron, induction, electrodynamic, moving-coil, and other types.

4-2. Moving-iron Instruments

A moving-iron instrument of the attraction type (Fig. 32) consists of a fixed coil *1*, a movable steel vane *2*, an aluminium damper sector *3*, both seated on a shaft *4*, a pointer *5*, and a spiral spring *6*. The damper *3* is placed in the field of a permanent damping magnet *7*. The shield *8* screens the magnetic field of the damping magnet.

When the current to be measured is passed through the coil, the coil is magnetized and attracts the vane against the opposing force

of the spring, thereby turning the shaft and pointer through an angle. The greater the current passed through the coil, the larger the deflection of the pointer. As it turns, the shaft rotates the aluminium
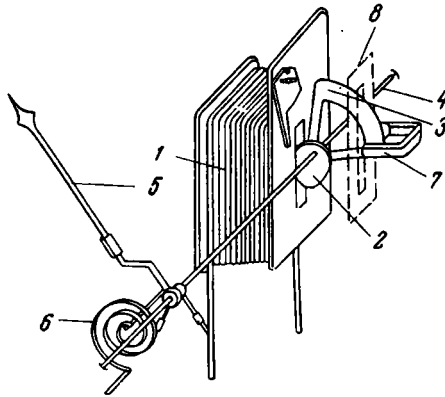

Fig. 32. Diagram of a moving-iron instrument

sector through the field of the permanent damping magnet, and eddy currents are induced in the aluminium disc, and interact with the permanent magnet field to produce a braking torque which damps the swings of the pointer. This type of instrument is suitable for measurements in both d.c. and a.c. circuits.

4-3. Electrodynamic Instruments

The movement of an air-cored electrodynamic instrument (Fig. 33) consists of a fixed coil *1* and a movable coil *2*. The movable coil *2* is pivoted within the fixed coil on the same shaft with a pointer, the vane of an air-type damper, and two spiral springs *3*. The springs lead the current into and out of the movable coil and also supply the controlling force. The coils are connected in series. If the instrument is intended for the measurement of large currents, the movable coil is wound with fine wire and connected to the circuit by means of a shunt. When the current to be measured flows through the movable coil, the latter turns together with the pointer through an angle dependent upon the value of the current.

The air damper in this instrument consists of a chamber *4* within which a light vane *5* attached to the moving system moves. When the moving system turns on its pivots, pressure of the air in front of the vane in the chamber is increased while the pressure behind it drops. The resultant difference in pressure retards the movement of the vane and damps the swings of the moving system.

Iron-cored electrodynamic instruments (Fig. 34) are a modification of electrodynamic instruments. The fixed coil in one type of such an instrument consists of two halves *1* and *2* arranged on two limbs of a steel magnetic core *3* between the ends of which is placed a fixed steel core *4*. Within the air gap is arranged a movable coil *5* which, together with a pointer, is attached to a pivoted shaft. When the current to be measured energizes the fixed coil, a magnetic field
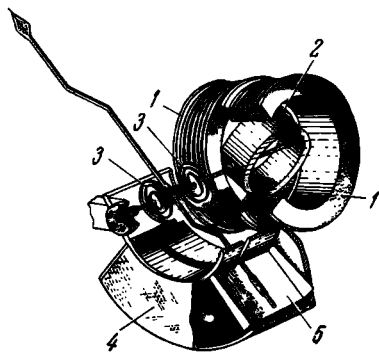

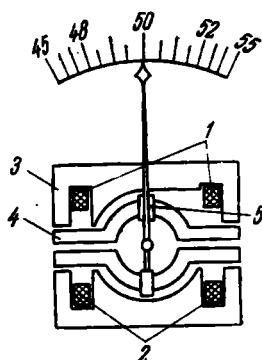Fig. 33. Diagram of an air-cored electrodynamic instrument

Fig. 34. Diagram of an iron-cored electrodynamic instrument

is produced in the air gap, proportional to the unknown current. The interaction of the current traversing the movable coil with the magnetic field in the air gap results in a certain angular displacement of the coil and its pointer.

The use of iron cores leads to large magnetic fluxes and, consequently, to a higher working force, or torque, acting upon the moving system of the instrument. Therefore, a very rugged moving system can be built.

Air-cored and iron-cored electrodynamic instruments are suitable for measurements in both d.c. and a.c. circuits.

4-4. Induction Instruments

Fig. 35 shows the arrangement and connection of an induction watt-hour-meter. It has two electromagnets *1* whose coils are connected in series with the load, and *2* whose coil is connected in parallel with the load. Between the poles of the electromagnets an aluminium disc *3* is arranged so that it can rotate together with a pivoted shaft on which a worm *4* is also seated. The latter is geared to a register. When the electromagnets are energized with alternating currents

I_1 and I_2, alternating magnetic fluxes are produced; they cut the aluminium disc and therefore induce eddy currents in it. The eddy currents interact with the magnetic fluxes and cause the disc to rotate, thereby actuating the registering mechanism. The disc rotates
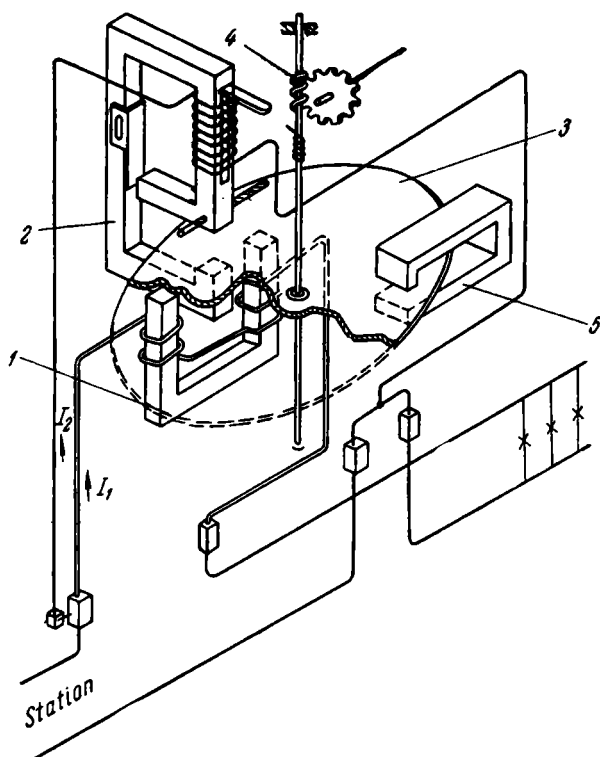
Fig. 35. Diagram and connection of an induction
watthour meter

in the field of a permanent magnet *5* which serves as a damping device. When the load is switched off, the disc quickly stops under the braking action of the currents induced by the field of the permanent magnet. Induction instruments can only be used on a.c. circuits.

4-5. Measurement of Current and Voltage

Electric current is measured with ammeters which are connected in series with the circuit. Ammeters must be of low resistance so that they will not alter the current to be measured. For this reason

their coils are wound with wire of sufficiently large diameter to carry the maximum measured current without excessive voltage drop. Ammeters may be of the moving-iron, electrodynamic and ferrodynamic type. Ammeters of the last two types have their fixed and movable coils connected in parallel. Switchboard ammeters are mainly of the moving-iron class.

Voltage is measured with voltmeters which are connected in parallel with or across the circuit, and must therefore have a high resistance. Owing to this, voltmeter coils are wound with a large number of turns of fine wire.

Voltmeters may be of the same types as ammeters. In electrodynamic voltmeters both coils are connected in series.

4-6. Measurement of Electric Power and Energy

A.c. power is measured with wattmeters which may be of the electrodynamic (or ferrodynamic) and the induction type. In an electrodynamic instrument, the coils are not connected to each other, and four terminals are provided on the case. The fixed-coil terminals are marked "*I* *" and "*I*", those of the movable coil "*V* *" and "*V*". The fixed coil is called the current coil and is connected in series with the load. The movable coil is called the voltage coil and is con-
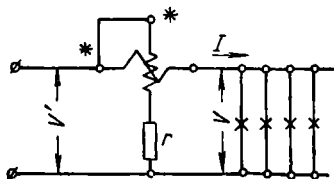
Fig. 36. Connection of an electrodynamic wattmeter
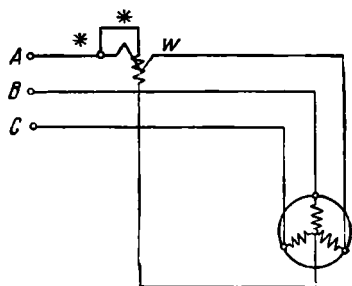
Fig. 37. Connection of a wattmeter for balanced loads with brought-out neutrals

nected in parallel with the load (Fig. 36). The terminals marked with an asterisk (*) are made common with a jumper and connected in the circuit on the supply side. These terminals are sometimes called "generator terminals". Improper connection of these terminals will cause the wattmeter pointer to deflect in the wrong direction.

The magnetic field produced by the current coil is proportional to the load current; that of the voltage coil is proportional to the circuit voltage. Both magnetic fields interact so that the angle of deflection of the moving system is proportional to the true or active power in the load.

In design the induction wattmeter does not, in principle, differ from the watthour-meter. In a wattmeter the angle through which the aluminium disc turns is directly proportional to the power taken by the circuit. Indication is given by a pointer attached to a

Fig. 38. Circuit for power measurement by the three-wattmeter method
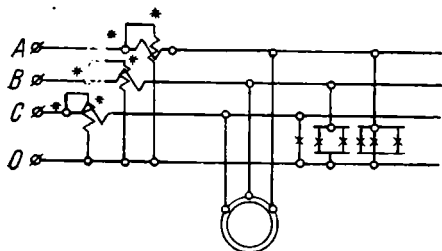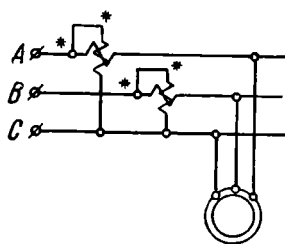
Fig. 39. Circuit for power measurement by the two-wattmeter method in a three-wire circuit

pivoted shaft which carries the disc and a spiral restraining spring. As in the case above, one of the coils is connected in series and the other in parallel with the load.

For a three-phase circuit carrying a balanced load, the active power will be three times the readings of a single-phase wattmeter. Fig. 37 shows the connection of a single-phase wattmeter to a balanced three-phase circuit where a neutral wire or line is available. In unbalanced four-wire three-phase circuits (supplying both lighting loads and three-phase motors) it is necessary to use three wattmeters. The total power in the circuit will then be the sum of the three wattmeter readings (Fig. 38).

The power in a three-phase circuit without a neutral wire, whether balanced or unbalanced, can be measured with either two wattmeters or one two-element wattmeter (Fig. 39). The latter instrument comprises two single-phase induction wattmeters which are combined to act upon a common aluminium disc. The indications of both wattmeters are mechanically added to provide a direct reading.

Energy consumption is measured by the watthour-meters (or simply meters) discussed earlier. Meters are connected to single-phase and three-phase circuits in the same way as wattmeters.

The power factor of circuits is measured with indicating instruments called power factor meters or indicators. They are of the electrodynamic or moving-iron type. An electrodynamic power-factor meter has one fixed coil for series connection with the circuit and two movable coils. One of the latter is connected in series with a resistance, thus making up a resistive branch; the other is connected in series with an induction coil, thereby constituting an inductive

branch. The two branches are arranged in parallel and carry current proportional to the circuit voltage. In such an instrument the pointer will deflect in proportion to the phase angle between the current and the voltage of the circuit, and the scale is calibrated directly in values of power factor (cos φ).

A moving-iron power-factor indicator consists of two fixed coils, one connected in series with the circuit (the field coil) the other connected in parallel with the circuit (the magnetizing coil). The moving system comprises a pivoted shaft which carries a two-vane magnetic core and a pointer. The restraining force is provided by a spiral spring.

The frequency of an alternating current in a circuit may be determined in several ways. Direct indication of the frequency is given by frequency meters. The most common is the vibrating-reed type. It has a series of thin strips, or "reeds", each rigidly fastened to a cross bar at one end and free to vibrate at the other. The reeds are placed in the field of an electromagnet which is energized from the a.c. circuit under measurement. Each reed is adjusted, or tuned, to a definite natural frequency of vibration within the desired range of measurement. When the electromagnet is energized, its magnetism alternates with the frequency of the supply and the reed having the natural frequency closest to the frequency of the field will vibrate most appreciably. The ends of the reeds are turned up and painted white, so that the reed vibrating most will be identified by a white band or blur. The frequency can then be read from the instrument by observing the scale mark opposite to the white blur. A frequency meter is connected to the circuit in the same way as a voltmeter.

4-7. Shunts, Voltmeter Multipliers, Current and Voltage Transformers

Ordinary electrical instruments are suitable only for measuring low currents, voltages and power. However, it may be necessary to use the same instruments for measuring large currents, high values of power and high voltages. The range of an ammeter can be extended by connecting the instrument in parallel with a low resistance (a shunt). Then only a small fraction of the circuit current will flow through the instrument, the remainder flowing through the shunt. The instrument thereby carries a definite portion of the current while its pointer indicates the value of the total current.

Usually shunts are selected so that 0.9 or 0.99 of the total current will flow through it. Then the instrument will carry 0.1 or 0.01 of the current being measured.

The range of voltmeters can be extended by connecting a non-inductive resitance, called the voltmeter multiplier, in series with the

instrument. A multiplier is selected so that it will drop the main portion of the circuit voltage. Then the voltage drop across the instrument will be negligible.

Fig. 40 shows the connections of an ammeter with a shunt, and a voltmeter with a multiplier to a circuit.

In a.c. circuits, the range of instruments can be extended by means of instrument (current and voltage) transformers.
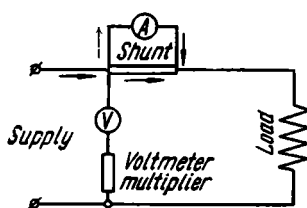

Fig. 40. Measuring instruments connected through a shunt and a voltmeter multiplier
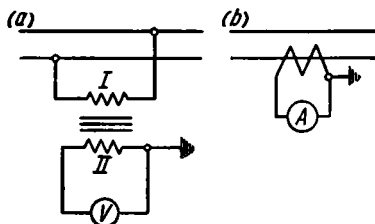

Fig. 41. Instruments connected through voltage and current transformers:
(a) voltmeter; (b) ammeter

In the case of voltage measurements, the primary *I* of a voltage transformer (Fig. 41a) is connected in parallel with a high-voltage circuit, and the voltmeter is connected across the secondary (or low-voltage) winding *II*. Voltage transformers are so designed as to develop a standard voltage of 100 V on the secondary side.

One side of the secondary in the voltage transformer must always be earthed in order to protect operating personnel from shock hazard. This hazard may be due to a high-voltage potential developing on the low-voltage side after a breakdown or fault in the insulation between both windings. The tanks and cores of voltage transformers must also be earthed.

A current transformer (Fig. 41b) extends the range of current measurements. Its primary is connected in series with the circuit in which the current must be measured.

A current-transformer primary consists of one or a few turns of heavy wire. The secondary, connected in series with an ammeter, contains a large number of turns of fine wire.

Current transformers are designed so that a current of 5 amperes will flow through the ammeter when rated current is passed through the primary. Thus, if the nameplate of a current transformer reads "200/5", it means that when the primary carries 200 A, the ammeter will read a secondary current of 5 A.

The secondary of a current transformer must always be connected to an ammeter or other loads, or be short-circuited. If left open-cir-

cuited, a high voltage dangerous to life can appear across the second-
ary. Furthermore, the high voltage may also damage the secondary
winding insulation.

As is the case with voltage transformers, one of the secondary ter-
minals of a current transformer must always be earthed for safety
from shock hazards.

Example 16. The current coil of a meter is connected to a 200/5-ampere
current transformer and its voltage coil is connected across a 6000/100 V
voltage transformer. After a certain interval of time, the meter registered 10 kWh.
Find the actual energy consumption in the circuit during the same period.

The actual energy consumption will be equal to the product of the current
and voltage ratios of the respective instrument transformers multiplied by
the registered energy.

The ratio product:

$$\frac{200}{5} \times \frac{6000}{100} = 2400.$$

The actual energy consumption is

$$10 \times 2400 = 24,000 \text{ kWh.}$$

4-8. Measurement of Insulation Resistance

Insulation resistances are measured with an instrument called a
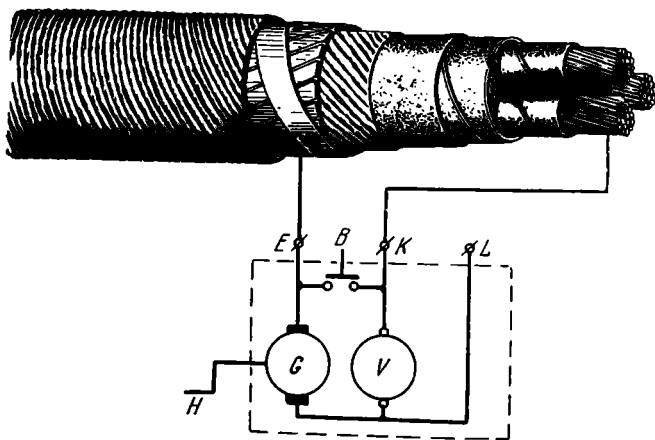megohmmeter (megger, insulation tester).

Fig. 42. Circuit for insulation resistance measurement

A megohmmeter (Fig. 42) comprises a small generator G hand-
driven with a crank handle H, and an indicating instrument V. The
instrument is connected in series with the resistance to be determined
and has its scale calibrated in both volts and ohms.

When cranked, the generator supplies the measuring circuit with a direct current.

In order to determine the insulation resistance of a cable conductor to earth, the terminal *K* of the megohmmeter is connected to the conductor under test and the terminal *E* to earth (the steel cable armour); the button *B* is held down, and the handle is cranked at a uniform speed of about 2 to 3 revolutions per second until the pointer deflects to the end of the scale. The button is then released while the handle is being still turned at the same speed. The final position of the pointer gives the value of the insulation resistance in ohms or megohms.

If it is necessary to find the insulation resistance of one conductor with respect to another, one conductor is connected to the terminal *K* and the other conductor to the terminal *E*. The measurement is taken as before.

The terminals *K* and *L* in this instrument are used for voltage measurements.

Where the insulation resistance of a motor winding to earth must be measured, a lead from an end of the motor winding is connected to the terminal *K* and the terminal *E* is connected to the motor frame. A low reading is an indication that the insulation has been damaged, or that the winding is in contact with the motor frame. Electrical safety rules (or codes) require that the insulation resistance should not be lower than 1000 ohms per volt of rated voltage. For example, the insulation resistance of a motor rated for 380 V should not be less than 380,000 ohms.

When a check on a cable circuit with a megohmmeter shows that one of the phases is faulted to earth, it is necessary to localize the fault. If the circuit consists of several parts, each part should be tested separately by consecutively switching out one section after another.

It must be always remembered that no circuit or motor shall be checked for its insulation resistance while energized.

Chapter V

FEATURES, OPERATION AND MAINTENANCE OF COAL MINING MACHINERY

A. CONVEYOR INSTALLATIONS

Haulage of coal from the faces to a gate roadway is accomplished by scraper-chain face conveyors.

At one time shaker conveyors were employed for this purpose, but they have been almost entirely superseded by scraper-chain conveyors.

Further transport of the coal along gate and gathering-haulage roadways, on inclines, etc., is done with belt conveyors.

5-1. Scraper-chain Conveyors

All scraper-chain conveyors can be divided into two groups:

(1) haulage conveyors used only for transporting the coal;

(2) integrated conveyors operating in conjunction with continuous-mining machines, coal ploughs or cutter-loader machines.

Apart from coal transport, integrated conveyors are also employed for coal getting during blast-loading operations, serving either as a track for a coalcutter, cutter-loader or other coal mining machine, or as a support for a coal plough.

In a scraper-chain conveyor coal is moved along a metal trough by scrapers or flights attached to a travelling chain. The latter is driven by an electric motor through a reduction gear and a drive sprocket. Coal can be handled by this type of conveyor both in horizontal and inclined headings.

In the Moscow coal fields scraper-chain conveyors are placed in tandem with belt conveyors in gathering gate roads where the gate conveyor line must be extended or shortened, as the case may be. Employment of a scraper-chain conveyor in such cases is desirable because it is much simpler to extend or shorten than a belt conveyor.

The working and return strands of these conveyors may be ar-
nged either one above the other, or side by side. It is also possible
have an arrangement in which the working strand is in a horizon-
td plane while the return strand is in an inclined plane.

These conveyors may be single-chain, double-chain and triple-
chain.

According to the way the conveyors are moved and set up in a new
position, they are distinguished as sectional conveyors intended for
manual advance, and portable or mobile conveyors for mechanical
moving or flitting. Portable or mobile type conveyors can, in turn,
be articulated or flexible, which can be snaked over as the coal cut-
ting-loading machine advances, and fixed conveyors which are flit-
ted as a whole.

A relevant U.S.S.R. standard divides scraper-chain conveyors
into four types: CK(SK), single-chain conveyors with cantilever
scrapers or flights and with both runs arranged in one plane (side by
side); C(S), single-chain conveyors with the working run above the
return run; CP(SR), double-chain sectional conveyors; and CΠ(SP),
double-chain type conveyors.

The numeral in the type designation of a conveyor indicates the
width of its trough. Thus, the designation CΠ-63 (SP-63) indicates
that the conveyor is double-chain, mobile, with a trough 63 cm wide.

Scraper-chain Conveyors for Seams up to 0.8 m Thick

CKT_2-6м (SKT_2-6m) and CKT_3-6(SKT_3-6) Scraper-chain Conveyors.
These conveyors have low troughs. They can take coal from the thin-
nest of measures ranging from 0.45 to 0.8 metre. Both of the open,
single-chain strands run in one horizontal plane; one serving to
transport coal away from the face, while the other (the return strand)
can be used to carry prop wood to the face.

The Driving Head. The driving head is set up at right angles to
the conveyor trough in the tail end of the conveyor, so that it is at
the head end of the face. The unit (Fig. 43) consists of a motor *1*,
gear box *2*, bedplate *3* and coupling *4* covered by a guard. There are
lugs on both sides of the bedplate for the attachment of conveyor
sections. The gear reduction consists of two pairs of helical gears and
one pair of bevel gears. The final (vertical) shaft of the gear box car-
ries the drive sprocket of the chain. For overload protection, the
gear reduction incorporates a disc-type friction clutch. It operates
as follows. One of the intermediate gears is free, to turn on its shaft;
to it is attached a drum with internal projections. Keyed to the same
shaft is a hub provided with projections on its outer surface. The
drum engages with a set of outer discs, while the hub engages with
a set of inner discs which interleave the outer discs. The pressure

between the discs can be adjusted at will with a screw. The clutch
thus transmits the torque from the motor via the gear and discs to
the shaft. As the pressure between the discs increases, the torque
applied to the shaft also increases. The clutch is factory-adjusted
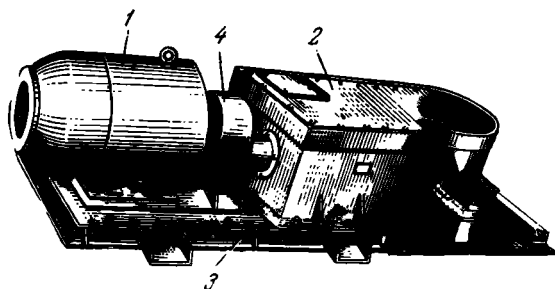for a maximum chain pull of six tons. It is not advisable to increase

Fig. 43. Drive head of CKT (SKT) conveyor

the disc pressure any higher, because an overload may stall the mo-
tor.

The bedplate of the driving head is also fitted with a chain-link
lifter which is a fork embracing the drive sprocket teeth on both
sides. It makes the chain links rise and run properly off the sprocket.
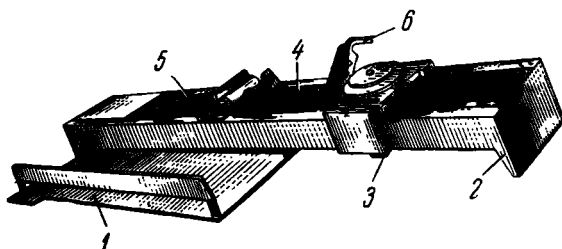
Fig. 44. Take-up head of CKT (SKT) conveyor

The Tension-end or Take-up Head. The tension-end or take-up
head (Fig. 44) is set up on a gate-end bench and comprises a frame *1*,
breast plate *2*, tension sprocket *3*, tensioning mechanism *4*, handle *5*
and ratchet *6*. The tension sprocket is keyed to its shaft so that the
tension sprocket assembly can be shifted along the frame by the ten-
sioning mechanism which consists of a screw and nut. The screw is
connected to the tension sprocket assembly, while the nut is restrict-
ed from linear movement by a set of stops. Rotation of the nut
causes the screw to move forward or backward, thereby shifting the
tension sprocket assembly as required for adjusting the scraper-chain

tension. The nut is rotated through a worm or ratchet drive mechanism. The worm drive is used for final tension adjustment (when the tension force is large). Preliminary tensioning, when the slack is being taken up in the chain, is done with the ratchet. In the latter case the nut remains stationary while the ratchet mechanism turns the screw to make it move in the required direction.

Conveyor sections. The CKT$_2$-6M (SKT$_2$-6m) and CKT$_3$-6(SKT$_3$-6) conveyors differ only in their trough sections. Both are designed with extra-wide sections to permit supports to be set up between the transport and return strands.

The CKT$_2$-6M (SKT$_2$-6m) conveyor is intended for operation in conjunction with an УKT-2M (UKT-2m) or KЦT (KTsT) cutter-loader. The latter is mounted with its skids on the conveyor trough for which reason the conveyor sections are of robust design and fabricated from steel channels for greater stiffness. The trough of the CKT$_3$-6 (SKT$_3$-6) conveyors is made much lighter and is of pressed construction.

The trough sections are joined together by square-section longitudinal bars at one end and eyed lugs at the other.

The Scraper Chain. The scraper chain consists of forged links and flights, the latter serving also as links of the chain. To avoid possible loss of separate chain links during repairs or move-ups, the chain sections are detachable only in lengths of 2720 mm. The chain must be assembled in the trough so that the flights travel with their smooth wall forward.

The KC-10 (KS-10) conveyor differs from the CKT-6 (SKT-6) conveyor in that it has a greater capacity and greater length. Its drive unit is set up alongside with and not at right angles to the trough and is placed between the transporting and the return strands. If the conveyor is very long, two drive units are used, one at each end. KC-10 (KS-10) conveyors are manufactured in five sizes. Various trough sizes can be obtained by fitting auxiliary sideboards on one or both sides of the trough.

KC-10$_4$(KS-10$_4$) and KC-10$_5$ (KS-10$_5$) conveyors are designed for headings with undulating floors and are fitted with guides for the chain and flights.

Like the CKT-6 (SKT-6) conveyors, the KC-10 (KS-10) conveyors are of sectional construction.

The KC-15 (KS-15) conveyor has two chains and is of portable articulated construction. Its drive unit incorporates a fluid coupling. When the oil in the coupling reaches an excessive temperature, the conveyor is automatically switched off.

KC-15 (KS-15) conveyor sections are joined together by bolts and can be turned from 3 to 4 degrees relative to one another in a horizontal and a vertical plane.

The sections in this conveyor are made with pressed sideboards and from 8 mm steel bottom plates. The section ends are heat-treated to make them more resistant to wear.

The KC-15 (KS-15) conveyor can be set up with several drives, from one to four, to suit the length of the coal face and may have two drive units installed at each end.

The КСТИ-20 (KSTI-20) conveyor is of portable articulated design and has two chains. It is intended for delivering coal from faces in flat and gently-inclined seams with an undulating floor, and up to 0.8-metre thick. Its troughs are fitted with chain guides on both sides.

The trough consists of upper and lower members connected together by means of vertical bars and drilled plates. The take-up head is of the screw type.

The sections of this conveyor can be turned from 3 to 4 degrees relative to each other in both a vertical and a horizontal plane.

The CKP-11 (SKR-11) scraper-chain conveyor is of sectional construction and is fitted with one chain. Since the direction of chain travel can be reversed, it may be used to transport prop wood from the bottom gate to the coal face. The conveyor (Fig. 45) consists of a driving head *1* provided with a tripper, intermediate sections *2* which are set up between the drive and take-up heads, a take-up head *3*, a chain *4* and flights.
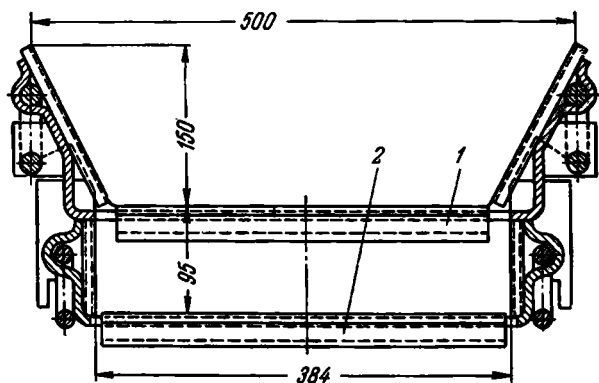

Fig. 46. Intermediate section of CKP-11 (SKR-11) conveyor

The intermediate sections fit together to form the trough. Each section (Fig. 46) consists of an upper *1* and a lower *2* trough member. The lower members are manufactured from 3 mm steel sheet and are joined together by a lock. The lock consists of a hook bent from round stock and fastened to one end of a trough member; it engages

a clip at the end of the trough member just ahead of it. The Svet Shakhtera Works manufactures CKP-11 (SKR-11) conveyors with interchangeable, i.e., identical upper and lower trough members.

The take-up head is a stationary framework anchored by jack props set up on its feet. It accommodates a moving frame which mounts the tensioning sprocket and its shaft, and also serves as a support for the take-up pan. The tensioning sprocket is keyed to its shaft which rotates in antifriction bearings. The slack of the scraper chain is taken up by shifting the moving frame in the stationary framework by a screw mechanism operated through a pair of bevel gears.

The CKP-20 (SKR-20) single-chain reversible conveyor is intended for transporting coal from faces with seams over 0.8 metre thick and for joint operation with Donbass or Gornyak cutter-loaders.

It is an improved version of the CKP-11 (SKR-11) conveyor and employs a more powerful driving head and a motor-driven ratchet take-up; its chain runs at a higher speed. The upper and lower trough members are interchangeable.

The KCPП-1A (KSRP-1A) single-chain, sectional conveyor has plow-shaped flights and pans—a feature which permits the trough to be made much narrower. The return strand of the chain is run back under the side of the conveying trough and slides directly over the floor. The upper and lower trough members are built integral. On the loading side the side of the trough is made higher.

The CTP-30 (STR-30) and CTP-30мy (STR-30mu) double-chain sectional conveyors are intended for transporting coal from faces in gently-inclined seams over 0.9 metre high. They are used in conjunction with cutter-loaders or for direct loading by shot firing. When the latter method is used, the conveyor is moved directly up against the face. If a cutter-loader is used, it is hauled along the face on the frame of the conveyor. Owing to this, the conveyor trough is designed to have adequate strength and rigidity. The trough sections of these conveyors are provided with detachable sideboards. No rigid connection is used between the take-up head and the trough, and the head is fitted at its sides with tensioning drums adjusted by a hand-operated worm gear.

The KC-9 (KS-9) double-chain conveyors are built in four modifications: $KC-9_1$ ($KS-9_1$) and $KC-9_2$ ($KS-9_2$) which are mechanically-moved, articulated or snaking units, and $KC-9_3$ ($KS-9_3$) and $KC-9_4$ ($KS-9_4$) which are sectional units intended for manual move-over.

In the first two articulated-type models the conveyor sections are joined by bolts which are not tightened fully and thus leave gaps for setting up the trough in a broken line, in a horizontal and a vertical plane. The driving head gear box in the KC-9 (KS-9) conveyor is placed between the bottom and top chain strands and is connected

to the drive motor through a fluid coupling whereby the conveyor is protected against breakdowns due to heavy overloads.

The CTC-3 (STS-3) and CT-6 (ST-6) scraper-chain conveyors have one chain and are of sectional construction. The CTC-3 (STS-3) conveyor drive incorporates a compact worm gear reduction, while the CT-6 (ST-6) conveyor drive has a spur reducing gear.

Both of these conveyors use troughs consisting of trapezoidal upper members while their lower members are open at the bottom. The conveying strand of the chain travels without any guides. Therefore, when the floor is uneven the trough must be levelled by means of boards.

The KC-2 (KS-2) double-chain conveyor. The trough in this conveyor can deflect through up to five degrees in a vertical plane and up to one degree in a horizontal plane, and so it is easy to adapt to an undulating floor.

This conveyor has its drive motor and gear box set up between the upper and lower chain strands. For protection against overloads, the drive unit incorporates a disc-type friction clutch.

The drive is fitted with a ratchet mechanism with the aid of which the drive motor can be used to tension the scraper chain. The drive unit can be moved by means of the rope drum mounted on the drive shaft and a set of snatch blocks on the drive unit frame.

In comparison with CTC-3 (STS-3) and CT-6 (ST-6) conveyors, the KC-2 (KS-2) model has a number of advantages:

(a) It can be adapted to the changes in the lay of the floor;

(b) It has a loading height of 183 mm, whereas the CT-6 (ST-6) and CTC-3 (STS-3) conveyors have a loading height of 286 mm;

(c) The conveying capacity and the length of haulage of the KC-2 (KS-2) conveyor is greater than that of other conveyors in use at the faces in the Moscow coal fields.

The KC-2 (KS-2) conveyor can be moved over by hand or by means of rack-type jacks without being taken apart.

Scraper-chain Conveyors for Thick Steeply Inclined Seams Mined by Horizontal Slicing (for shortwall faces, cross holes, cross entries, crosscuts)

The single-chain CK3м-1 (SKZm-1) conveyor has a rigid trough consisting of pressed pans, welded frames and steel-channel side members. Its take-up is of the screw type, with the screws located on the sides.

CK3-2 (SKZ-2) and CK3-3 (SKZ-3) single-chain conveyors employ the same type of trough as do CKP-11 (SKR-11) conveyors; the take-up head is the same as that used with CK3м-1 (SKZm-1) conveyor. In the CK3-3 (SKZ-3) conveyor, the drive and gear box are flange-mounted on the support framework.

Setting Up of a Conveyor. Before a conveyor is set up at a face it is first necessary to see that its roadway is straight enough and that all the coal, stone debris and prop wood have been cleared away.

The driving head is set up first. Next the bottom pans are put in place and joined together and the bottom strand of the scraper chain laid out. The upper pans are mounted, starting from the drive head, the upper strand of the scraper chain is set in place and joined to the bottom strand, and the chain then tensioned. When the scraper chain is being assembled, care must be taken that the heads of the pivot pins are fully seated in their nests in the outer chain links; the chain may otherwise jump off the drive sprocket. As each chain section is attached, it should be stretched so that all slack is taken up to avoid any trouble in tensioning the chain after the conveyor has been fully assembled. It is also important that the troughs have no sharp bends, as they may raise the chain in its operation. The troughing can be levelled with wooden boards slipped under the lower trough parts. The adjoining trough pans should fit closely to one another, with the edges meeting at the same level so that the chain can travel quietly, without bumping and jarring.

The floor immediately back of the take-up head should be levelled for 1 to 1.5 metres, to provide space for its movement when the slack of the chain is taken up. For ease of movement, a board should be placed on each side under the last section of the troughing and the take-up head.

In order to prevent the gummings from clogging the bottom chain strand when coal is loaded onto the conveyor, no sizeable gaps should be left between the upper and lower trough parts.

After the conveyor has been fully assembled, its drive head should be anchored and the chain stretched taut.

The drive head is anchored and protected from vibration by a strut laid on the floor behind and bearing against two struts set up on either side of the conveyor. Additionally, butting boards or blocks are jammed between the drive main frame and roof. Full-sized props must not be used for this purpose because the roof pressure may cause deformation to the drive unit.

The take-up head is anchored by two props set up behind and receiving the ropes from the take-up head.

The scraper chain is tautened by pulling back the take-up head by means of a worm mechanism. The tension must be increased at a steady rate, taking care that the take-up head is in correct alignment with the conveyor troughing. If the maximum possible travel of the take-up head has been used up before the chain is properly tautened,

it is necessary to push the take-up head forward, remove several links from the chain, and then retighten the chain.

Scraper chains should be tautened until all slack is eliminated on the run-off side of the drive sprocket.

The drive motor should be connected to the supply and tested for the direction of rotation immediately after the driving head has been mounted, and before the chain has been fully assembled. The reason for this is that the chains in a non-reversing conveyor must never be run in the backward direction. If the motor has not been checked for direction of rotation prior to the assembly of the scraper chain, this check must be performed by a jog start. If it proves nec-essary to reverse the motor, the connections of any two of the leads at the motor or magnetic starter must be interchanged.

After the conveyor has been installed, it should be given a trial run. Its gear box must be filled with oil and the drive chain lubri-cated before it is started. The conveyor should then be run without load for 10 to 15 minutes. During this interval the chain is watched to see that it correctly runs off the drive sprocket; the reduction gear, the coupling and the chain drive are also watched for proper operation. If no defects appear, the conveyor can be turned over to the face team.

When the coal faces are long, from two to three conveyors must be set up in tandem. In such cases the conveyor nearest to the gate loading point is to be set up first. The second conveyor is set up so that its delivery jib overlaps the take-up head of the first conveyor by 100 to 200 mm, with enough clearance left between the jib and take-up head for unrestricted travel of the return strand. For this, struts or blocks are placed under the driving head frame.

The electric motors which drive the tandem-arranged conveyors must be electrically interlocked so that the motor of the discharge-end conveyor starts first. When its starter closes, its interlock ener-gizes the starter of the second conveyor, and so on. When the conveyor motors are stopped, a reverse sequence takes place.

If a conveyor is to operate normally, a constant watch must be kept on its operation, any troubles eliminated as soon as they crop up, its parts lubricated at regular intervals, and all maintenance made to an approved schedule.

The mine workings in which conveyors are installed must always be maintained in a proper condition; the props should be set up with ample pitch and clearance from the conveyors to avoid any damage to the latter.

It is essential to keep the troughing straight and to eliminate any sharp bends which may cause the scraper chain to rise. The drive and take-up heads should be cleaned of gummings regularly. When a safety pin is sheared, the drive motor should be immediately

switched off, the cause for the overload located, the trouble remedied, and the pin replaced.

Chains have to be watched for missing flights; severed, lost or broken flights must be immediately replaced. Conveyors must be inspected every shift.

The points to be watched in inspection are adequate lubrication in the gear box and bearings, the condition of the chain and flights, and possible overheating of the motor, gear box and bearings.

After a conveyor has been set up on new roadway, its power cable must be inspected for possible damage, for proper connections to the motor, and earthing.

Advance of Scraper-chain Conveyors at Coal Faces. After a cycle of operation at a coal face is completed, the conveyor must be moved up. The procedure for this is as follows:

(1) The take-up head is freed from its anchoring and the chain slackened;

(2) The upper chain strand is taken down into sections from 4 to 5 metres long [for an CKT(SKT) conveyor, 2.4 to 2.73 metres long] and the sections are rolled into coils and carried over to the new position;

(3) The upper trough members are removed and placed on edge along the new path of the conveyor;

(4) The lower chain strand is carried over in the same way as the upper strand;

(5) The driving head is freed from its anchorings, detached from the trough and set up in its new position;

(6) The lower trough members are carried over to the new path, laid out and connected one after another, starting from the drive head;

(7) The take-up head is advanced and put up in position.

All further details concerning the assembly of the conveyors have been discussed earlier.

For a fast advance of a conveyor, its roadway should be cleared and prepared beforehand, and every man in the move-up team assigned a clearly defined job.

Lubrication. The gear reduction of a conveyor is lubricated with Grade Л(L) or T oil or a mixture of 50 per cent cylinder oil and 50 per cent oleonaphtha poured into the reduction gear casing so that the largest gear is covered to one quarter of its diameter. The bearings in the reduction gear casings are stuffed with grease when installed. The oil in the casing should be changed every four to six weeks. The old oil is drained through an opening in the lower part of the gear casing.

The head shaft bearings are lubricated with solid grease stuffed into the bearing housing and changed every four to six weeks, after

the head shaft parts have been cleaned. The overload clutch on the head shaft is oiled with an oil can at the beginning of each shift, the drive chain is lubricated with grease, also at the beginning of each shift. The tension sprocket bearings require grease for their lubrication, which is stuffed into their housing. Every three to four months the oil and grease is renewed after the parts have been cleaned.

The worm shaft bearings in the take-up head are lubricated with an oil can, the oil being added through the holes in the bearings.

The take-up shaft bearings are lubricated with grease by means of the grease cups screwed into the shaft ends.

Before a scraper chain is tautened, the worm reduction gear of the take-up head is lubricated by directly applying grease to the worm-wheel teeth.

Features of CKT (SKT) Conveyor Operation. The CKT (SKT) conveyor has certain particular features of operation. The drive unit can be installed either at the head end of the face or at the gate road end. When the seam is inclined, the drive head should be placed at the head end to prevent the scraper chain from gathering as it turns under and runs off the drive sprocket.

When the drive head is set up at the gate road end, there is no need for laying a flexible cable along the face road and for putting in an extra control panel box. However, in this case conveyor operation is made more difficult because the run-off side strand at the tail end has to travel uphill, owing to which the tension in the chain has to be increased.

Where gradients exceeding 12 degrees have to be overcome, it is not practical to place the drive head at the gate end.

A special U-shaped frame is used for setting up a drive head at a gate end.

The procedure for conveyor assembly depends upon where the driving head is to be placed. If it is to be set up at the head end of the face, the take-up head is first mounted at the gate end, following which the adapter section is attached and the intermediate troughing sections are put in place. The head end section is attached next (so that the chain throw-off device is on the return strand side) and the driving head is mounted.

When the conveyor troughing is fully assembled, the scraper chain is laid in place and, before the chain is tautened, the driving and take-up heads and the adapter section are anchored with the aid of a set of struts. The chain may now be pulled by means of the drive motor to take up the slack. This is done by bringing the chain ends out onto the conveying strand side at the driving head and jogging the motor while pulling the chain at the run-off end by hand until the opposite end starts to travel towards the take-up head. Following this,

the chain ends are joined together. The chain is finally tautened with the tensioning mechanism.

When the driving head is to be placed at the gate end, it is the first unit to be mounted and is placed on an underframe. Then all the consecutive troughing and connecting parts are attached. The head-end section is not needed at the drive side in this case, since it is set up forward of the take-up head.

An adapter section will not be required if the trough is made of steel channels, but when it is of pressed sheet steel construction an adaptor section must be inserted, both when the driving head is set up at the head or at the gate end.

Transport of Prop Wood. Props can be conveyed up to the face by the chain return strand in an CKT (SKT) conveyor and by the conveying strand [in an CKP-11 (SKR-11)].

The timbermen must always be warned before the props are run in to them.

Short props may be laid in the bottom of the trough between flights, but long props should be placed in the conveyor so that the forward end lays in the bottom of the trough. To avoid jamming a prop between the roof and the conveyor, it should be removed by its thick rear end.

Advance of an CKT (SKT) Conveyor at a Face. The advance should begin with the delivery end. For this, the chain is slackened and taken apart, the delivery end assembly detached, and advanced to the new position. After the driving or take-up head is advanced and attached to the delivery end, all the remaining sections may be moved up and the conveyor fully assembled.

The scraper chain should be taken down into sections of 2.4-2.72-metre length; the link pins should be stowed by the fitter in his tool kit bag to avoid their loss. When the conveyor is fully assembled, its chain can be tensioned and the conveyor given a trial run under load, during which the final adjustments of chain tension are made.

The section mechanic or the plant shift foreman and the coal shift foreman will accept the conveyor from the "advance" team only if it has been installed so that:

(a) all the trough pans have been locked and their mating ends fully coincide so that there are no abrupt steps to interfere with smooth travel of the scraper chain;

(b) the driving and take-up heads have been correctly mounted and accurately aligned with the line of travel of the scraper chain, since otherwise the chain may jump off the drive sprocket and run on the guard, causing damage to the links, flights and the chain;

(c) the trough is assembled straight;

(d) no one of the trough members have torn-off support lugs, since

in such a case an upper trough part may sink into the bottom trough and thereby lead to breakage of the chain return strand;

(e) the driving and take-up heads are reliably anchored so that they will not be moved off place or out of alignment.

The "advance" team may turn over a chain conveyor only after it has been fully assembled and has run satisfactorily for one hour under load.

Whenever a scraper chain is found to be worn, it must be changed and sent to the surface repair shop for overhaul and replacement.

5-2. Belt Conveyors

A belt conveyor comprises a flexible endless belt passing round a head or drive head drum at one end and round a tail-end or take-up pulley at the other. The top strand of the belt is supported by a set of carrying idlers, while the bottom strand is carried by return idlers or rollers. The belt is caused to move by the friction between the surface of the drive pulley or drum and the belt. The force of friction depends upon the belt tension, the coefficient of friction which is,
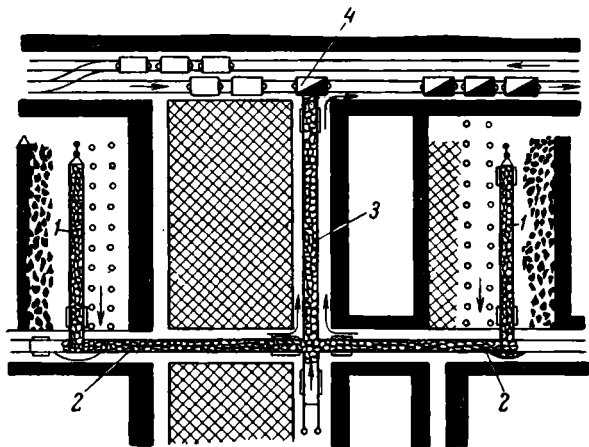

Fig. 47. Layout of belt conveyors in a gate road and an incline

in turn, dependent upon the material of the belt and the drum surface, and also upon the arc of contact with the drum. The arc of contact and, consequently, the pull at the belt, can be increased either by inserting a snub pulley or by using two drive drums. The load is usually carried by the top strand; the bottom or return strand runs idle.

In Soviet coal mines belt conveyors are used to transport coal in horizontal workings [РТУ-30 (RTU-30), КЛ-150 (KL-150) and КРШ-220 (KRSh-220) conveyors], in inclined shafts and headings [ЛКУ-250 (LKU-250), КРУ-260 (KRU-260) and КРУ-350 (KRU-350) conveyors], and in rise headings [КЛБ-300 (KLB-300) conveyors].

Fig. 47 shows a belt conveyor layout used in gate roadways and an incline. The scraper-chain conveyors *1* are set up along the coal faces from which they deliver coal onto belt conveyors *2* placed in the gate roadway. From the gate belt conveyors the coal is dropped onto the belt conveyor *3* which is set up on the incline and discharges into mine cars *4*. The paths of the coal are shown by the arrows.

Scraper-chain Conveyor Troubles, Their Causes and Remedies

Trouble	Cause	Remedy
Knocking in gear box	Worn gears and bearings	Replace gears and bearing
	Improper meshing due to poor assembly of gearing	Reassemble the gearing
	Dirty oil	Drain and refill with fresh oil
Gear box heats near bearings	Damaged bearing or lack of lubrication	Replace the bearing; add lubricant
Oil leaks from gear casing	Bad gasket at gear casing joint where tapered pinion-shaft is situated	Replace seals and gaskets
	Missing gasket under drain plug head	Put in gasket
	Inclined gear box	Level off drive head
	Crack in gear casing	Weld up crack
Scraper chain moves in jerks	Insufficient chain tension	Tighten the chain
	Jammed flights on bottom strand	Remove damaged flights
	Gummings on bottom strand or in take-up head	Remove gummings; look for loose joints in lower strand
	Flights catch at end pan as they come up into upper strand at take-up head	Do not leave prop trimmings or stubs in under-troughing when setting up conveyor
	Bad joints in undertroughing interfere with free movement of chain flight; bent flights in chain	Straighten bent sides of troughing; remove bent flights
Frequent shearing of overload safety pin	Overloaded drive head due to excessive length of conveyor	Maintain length within limits permissible for given model or type

Continued

Trouble	Cause	Remedy
Hot head shaft bearings	Damaged bearings Lack of, or dirty lubricant	Replace Add or change grease
Lubricant quickly leaks out of bearing	Poor gaskets under bearing caps; worn seals	Replace gaskets; remedy or replace seals
Scraper chain moves in jerks	Worn take-up sprocket Damaged or improperly assembled links in scraper chain	Replace sprocket or turn it through 180 degrees Replace faulty links; reassemble chain
Gear box and motor vibrate	Motor frame loosely bolted to base plate Bolts securing gear box and motor are loose	Tighten bolts Tighten bolts
Chain flights jam in undertroughing	Bent side walls in undertroughing	Straighten bent troughing
Scraper chain rises; a bed of coal appears on trough bottom	Low spots or hollows in troughing	Level off troughing with blocks placed under the sections
Slack scraper chain	Shifted anchor struts of holding ropes	Wedge anchor struts tight
Motor fails to start up when conveyor is fully loaded	Excessive load In an CKT (SKT) conveyor, loose friction clutch, discs slip	Shorten length or reduce load Increase pressure between clutch discs
When CKT (SKT) conveyor is started, its motor stalls	Overloaded conveyor; excessive pressure between friction clutch discs	Reduce load; decrease pressure between friction clutch discs
When a prop jams under the gear casing, friction clutch in CKT (SKT) conveyor fails to slip	Excessive pressure on friction clutch discs	Reduce pressure

The specifications of the belt conveyors mentioned above are given in Table 3.

In horizontal workings it is usual to install the PTУ-30 (PTU-30) conveyor.

The PTУ-30 (RTU-30) conveyor consists of a drive head and motor, a take-up pulley, cradles for idlers, side plates and a belt jointed with locking fastenings.

Table 3

Characteristic	Type of conveyor					
	РТУ-30 (RTU-30)	КЛ-150 (KL-150)	КРШ-220 (KRSh-220)	КРУ-260 (KRU-260)	КРУ-350 (KRU-350)	КЛБ-300 (KLB-300)
Maximum length, m	300	500	300	850-500 *	900-550 *	—
Conveying capacity tons/hr	80-120-180	150	220	260	350	300
Belt speed, m/sec	0.67-1-1.5	1.1	0.91	1.5	1.5	0.96
Belt width, mm . .	.700.	800	900	900	1200	900
Drive motor rating, kW	15; 21.5; 29	32; 25; 20	35	2×90	3×90	32

* The first figure stands for a heading gradient of 8 degrees, and the second figure for an 18-degree gradient.

The belt can travel at one of three speeds: 0.67, 1.0 or 1.5 m/sec. The motor and gear box may be mounted either on the left or right of the belt.

The idlers supporting the top or carrying belt strand, or the troughing idlers, are of the three-pulley type, while the return idlers are of the single-pulley type.

The conveyor drive head (Fig. 48) consists of a sectional frame *1*, gear box *2* with couplings *3* and *4*, two drive drums *5* and *6* carried by bearings *7* and *8*, a jib with an overhung unloading or delivery pulley *9*, idlers *10* and *11*, a wiper *12* for cleaning the belt, and an electric drive motor *13*.

In order that the drive head may be quickly taken apart, the motor is connected to the gear box, and the gear box, to the drive drum by sectional-type couplings. The overhung delivery pulley is detachable to allow it to be set up at any distance from the drive head. In this way, less noise will be heard from the drive, conveyor operation is made safer, and better conditions are created for maintaining the drive head.

The motor (Fig. 49) drives the rear drive drum *1* through a three-stage gear train. The first stage consists of two straight-tooth bevel gears; the second and third stages are two pairs of helical gear wheels. All the shafts revolve in roller bearings. The gear reduction casing is of cast iron and serves to support the shafts, to enclose and protect the gearing from dirt, and also to hold lubricating oil.
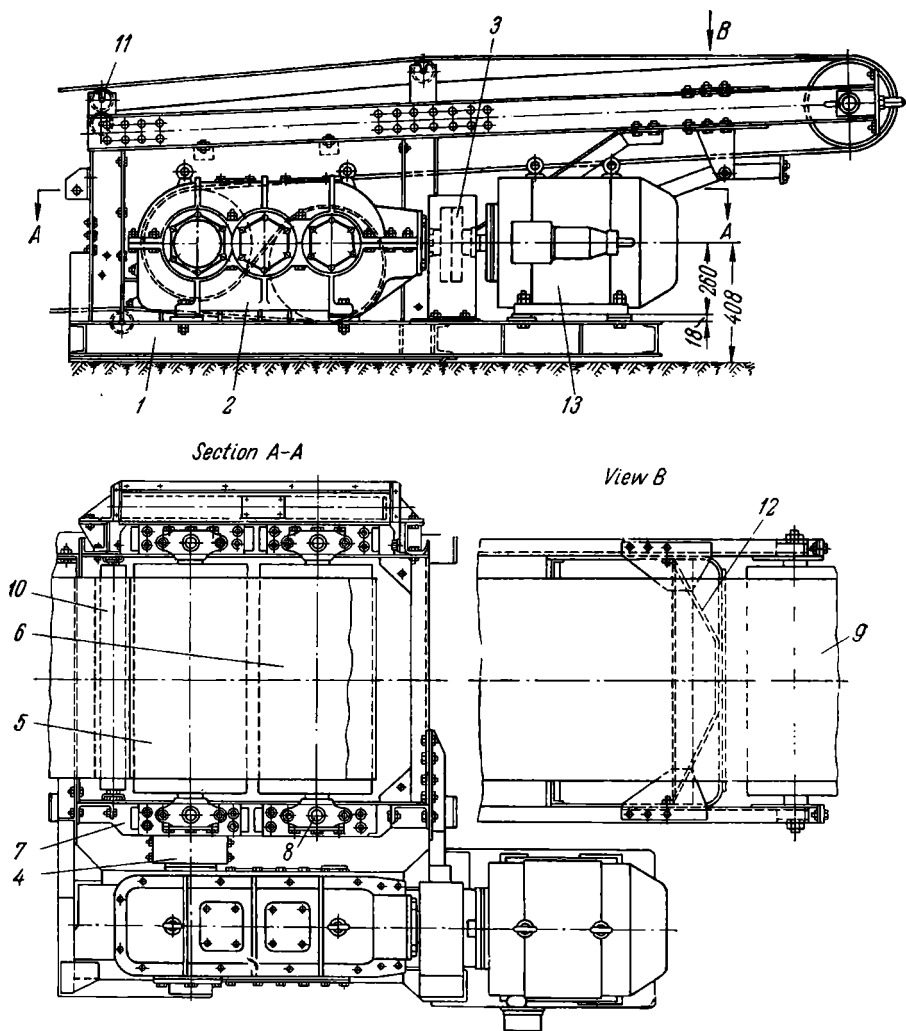
Section A-A

View B

Fig. 48. Drive head of the РТУ-30 (RTU-30) conveyor

The gearing and bearings are splash-lubricated as the gears rotate in the oil bath. An oil drain is provided in the bottom part of the gear casing. It is closed with a screw plug fitted with a gasket under its head.

The front drive drum *2* is actuated by the gears *3* and *4* which are enclosed in a separate weld-fabricated casing also serving as an oil bath.

The drive motor and the main gear box are mounted on a separate bedframe secured to the main frame by bolts.
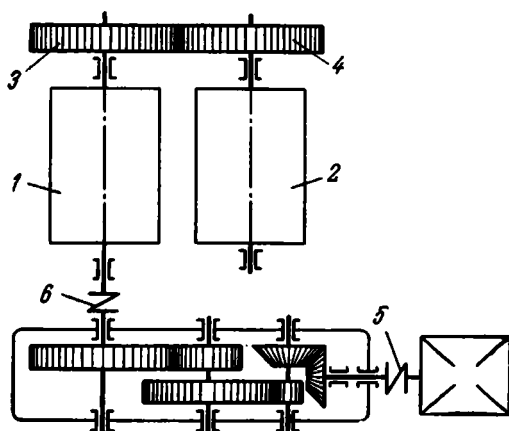
Fig. 49. Gearing diagram of the РТУ-30 (RTU-30) conveyor drive head

The flexible coupling *5* connecting the motor to the reduction gear consists of two halves, one seated on the motor shaft, the other on the gear box shaft. The two halves engage by means of steel drive studs fitted with rubber sleeve rings. On the output side, the gear box shaft is connected to the drive drum shaft by a chain coupling *6*. The latter consists of two sprockets, one fastened on the gear box shaft, and the other on the rear drive-pulley shaft. The two sprockets are coupled together by a plate-type roller chain member.

The drive-drum shafts are carried by spherical self-aligning ball bearings.

The belt passes round both drive drums; a total arc of contact of 480 degrees (Fig. 50) is thus obtained.

For convenience in discharge of the coal from the conveyor into a mine car or onto another conveyor, the delivery pulley is mounted on an extension jib of the frame. This jib consists of two steel channels fastened by bolts to the drive head frame and to the frame carrying the delivery pulley. Near the latter a wiper is mounted for clean-

ing the belt before it travels round the drive drums. The wiper is
a rubber strip attached to a pivoted and suspended frame and is
held against the belt surface by means of a weight.

The take-up head of the conveyor (Fig. 51) sets up the required pull
on the belt at the drive drums and limits the sag of the belt. It con-
sists of a metal framework *1*, drum or pulley *2*, belt cleaner *3*, ten-
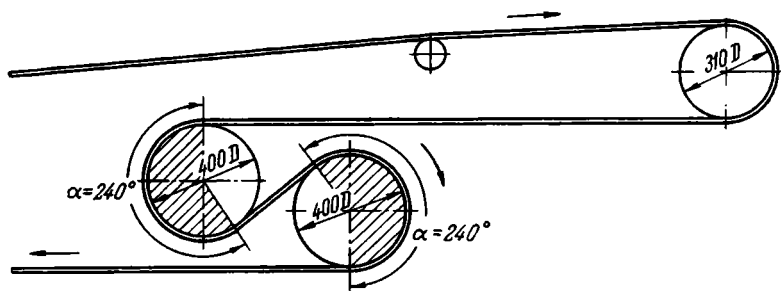sioning device *4*, feed funnel *5*, pulley cleaner *6* and guide roller *7*.

Fig. 50. Method of passing belt round drive pulley in PTУ-30 (RTU-30)
conveyor drive head

The take-up pulley *2* is carried by spherical self-aligning ball bear-
ings *8* bolted to the pulley discs *9*. The bearing caps have felt and
labyrinth seals *10*. Grease cups *11* provide the means for lubricating
the bearings.

The tensioning device incorporates worm gears and rope drums *12*
which take up the tensioning rope. The rope drums are cast integral
with wormwheel rings *13* and are free to turn on a shaft *14*. The self-
braking worm *15* is mounted vertically and is carried in bearings *16*
and *17*. It is operated by means of a handle *18* which acts through a
pawl and ratchet. The ends of the tensioning ropes are fixed on the
rope drums by clamps *19*. At the opposite ends the tensioning ropes
are tied either to props or to the frame of the next conveyor. To taut-
en the belt, the take-up head is shifted as a whole along the floor.
The belt cleaner *3* consists of two V-bent strips with a rubber strip *20*
fitted between them. The belt cleaner is held against the bottom belt
by its own weight.

The pulley cleaner device *6* consists of a trough *21* with a renewable
scraper *22*. The coal removed by the scraper is caught by the top belt
strand and carried over onto the trough from which it falls on the
bottom belt strand *23* where it is removed by the cleaner *3*. The fun-
nel *5* guides the coal onto the belt. It is a welded box with a square
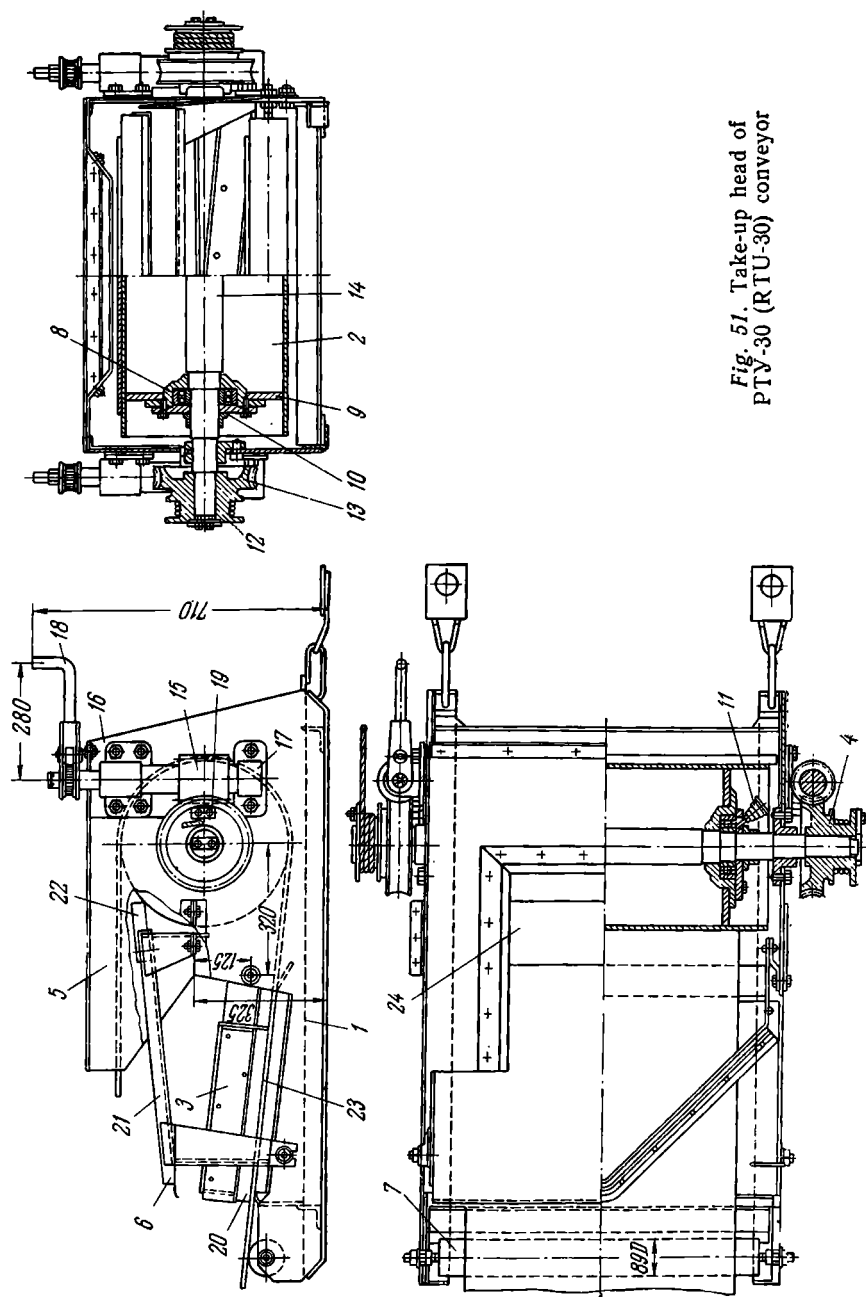opening *24*.

Fig. 51. Take-up head of
РТУ-30 (RTU-30) conveyor

Depending on the requisite length, each conveyor is assembled of a number of sections. Each section consists of cradles, idlers, and belt guards or purlins.

The cradles (Fig. 52) carry the top-belt and bottom-belt idlers. They are set up directly on the floor or on wooden foot boards and are spaced on 1400 mm centres. A cradle consists of a frame *1*, top-belt (or carrying) idlers *2*, a bottom-belt (or return) idler *3*, and a foot bracket *4*.

Bottom-belt idlers or rollers *3* have their shafts inserted in the side openings of the frame *1* and secured on each side with nuts. The foot brackets *4* have drilled holes through which the cradles are fastened to wooden sleepers or other types of bases. The belt guards are held in place with bolts *5* and nuts *6*; the nuts have levers *7* for tightening directly by hand.

The top-belt (or carrying) idlers are of the three-pulley type and are mounted in a common bracket. The middle pulley is horizontal, while the two-side or troughing pulleys make an angle of 20 degrees with the horizontal and are tilted 3 degrees forward in the direction of belt travel (to prevent the belt from sliding sideward).

The belt guards in each section are used to keep lumps of coal and stone from getting onto the return strand of the belt.

The guards are set up on the projecting ends of adjacent cradles and joined together to form a continuous box-like frame. The guards look like troughs mounted bottom up and have stiffeners welded along their edges. The stiffeners have slots for slipping the guards onto the studs in the cradles where they are secured by nuts. When there is no need for protecting the return strand, purlins are mounted in place of the guards. They tie up the cradles into a single rigid system. The purlins consist each of steel angles to which corner plates are welded at the ends. They are fastened to the cradles by bolts *5*.

PTУ-30 (RTU-30) conveyors employ rubber-lagged duck belts 700 mm wide and 11.5 mm thick. Belts of this type come in rolls up to 75 metres long. The separate sections are jointed by steel plates *1* (Fig. 53 *a*) placed on each side of the belt end and fastened by rivets *2*.

A better method of belt jointing uses wire hooks (Fig. 53 *b*). The hooks *3* are manufactured from steel wire and are arranged along the belt ends to make staggered connections in the belt material. Sets of these hooks come arranged on holder strips *4* so that they can be slipped over the end of the belt and compressed with a special vise. After both ends of a belt have been prepared in such a manner, the belt can be joined by slipping a thin steel pin *5* through the hooks. A still better way is to joint the belts by vulcanization.

The electrical equipment of these conveyors consists of a drive motor, starting apparatus and a feed cable.
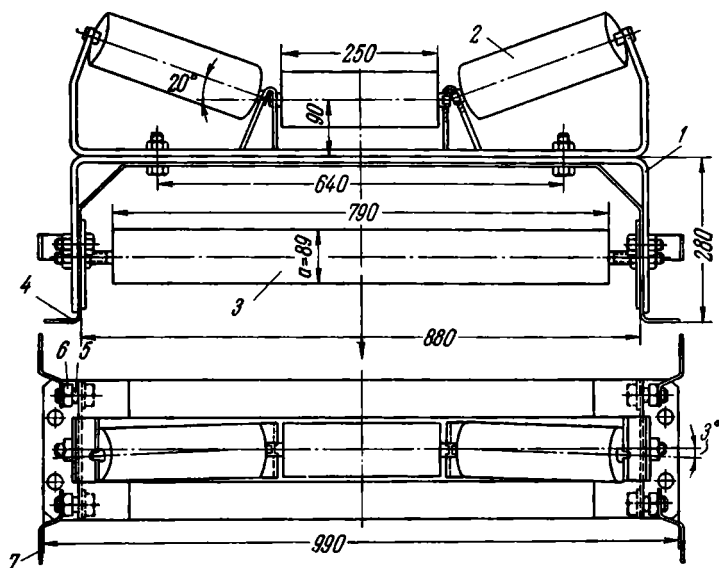
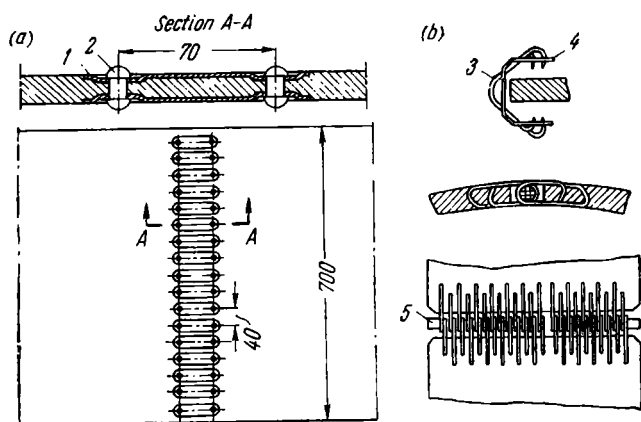*Fig. 52.* Conveyor cradle complete with idlers

Fig. 53. Conveyor belt joints:
a—joint with riveted steel straps; *b*—joint with wire hooks

To cope with a variety of conveyor lengths, belt speeds and conveying capacities, the drive heads of these conveyors will permit installation of motors of 15, 21.5 and 29 kW rating.

The КЛ-150 (KL-150) conveyor is a semistationary installation and is intended for coal transport both in horizontal workings and in rise headings and inclines with gradients up to 18 degrees. It may be 500 m long on gradients of 8 degrees and 300 m on gradients of 18 degrees. The drive head of this conveyor has two driving drums, each driven by a separate motor through a gear box and a fluid coupling. Both drums are rubber-lagged, which fact has increased the coefficient of friction and the pull of the drive by 92 per cent as compared with the РТУ-30 (RTU-30) conveyor, and so a snub pulley may be omitted.

The КРШ-220 (KRSh-220) conveyor is intended for coal transport in horizontal roadways or roadways where the gradient is small. In this conveyor, the belt is tensioned by two ropes wound on the drums of a hand-operated winch. The drive head has two drums driven by two electric motors, one of which is a reserve unit.

The КРУ-260 (KRU-260) conveyor is intended for installation in inclined workings of great length and can be up to 850 metres long on gradients of 8 degrees, and up to 500 metres long on gradients of 18 degrees. It uses a wire-rope reinforced belt. Its drive and delivery drums have rubber-lagged surfaces. The primary shaft of the gear reduction is fitted with a weight-loaded shoe-type electromagnetic brake released when the conveyor is started up.

The КРУ-350 (KRU-350) conveyor is identical in design with the КРУ-260 (KRU-260) conveyor, differing only in that its drive is more powerful, having three drive motors in place of two, and in that its belt is wider. Fluid couplings have been incorporated in the drum drives to ensure smooth start-up of the conveyor and equal distribution of the load between the electric motors. The high-speed shafts of the gear boxes carry brake pulleys and are fitted with band brakes to grip the pulleys as soon as reverse rotation begins. Thereby, the loaded belt is prevented from running downhill when the motors are switched off. Electromagnets release these brakes when the conveyor is started up and is in operation. To keep the belt clean, this conveyor incorporates a brush-type cleaner.

On gradients of 8 degrees, the conveyor can be 900 metres long, and on gradients of 18 degrees, 500 metres long.

The КЛБ-300 (KLB-300) conveyor is for use in rise headings and has its drive head placed at the uphill end. Its two drive drums are actuated by a single electric drive motor through a common gear box. The tension device is placed at the delivery (downhill) end. Owing to the self-braking planetary reduction gearing, no downhill movement of the loaded belt can occur when the electric motor is switched off.

The **КЛ-15 (KL-15) conveyor** is a semistationary installation for use in horizontal and inclined workings. Its drive head includes an automatic tensioning arrangement with a separate 0.9 kW motor for its operation. The belt of this conveyor is made of high-strength capron fabric (the Soviet brand of nylon). This conveyor is still in the experimental stage.

The **КЛЦ-1Ц (KLTs-1Ts) belt-chain conveyor.** Belt conveyors of extreme length require belts of high tensile strength. However, this necessity can be obviated by using an ordinary-strength belt to carry the load, while the tension and pull will be taken by wire ropes or chains—a combination known as belt-chain conveyors. Conveoyrs of the belt and wire rope class employ a rubber belt supported by strong wire ropes. Such conveyors find application in opencast mining.

In the КЛЦ-1Ц (KLTs-1Ts) conveyor the belt is also used only to carry the load, while the pull is taken by a pin-bushing-roller chain with a pitch of 200 mm. From the chain the pull is imparted to the belt by the dish plates attached to the inner and outer links of the chain by means of split pins. On the outside the dish plates are lagged with fluted or checkered rubber for a better grip on the belt.

The КЛЦ-1Ц (KLTs-1Ts) conveyor will transport coal in inclined workings with gradients up to 18 degrees, and also in horizontal workings of any practical length. Where the workings are of extreme length, the driving head is supplemented by intermediate drive units. The head-end drive includes an electric motor, fluid coupling, brake unit, gear box and a belt cleaner.

Care of Belt Conveyors

To maintain a conveyor in working condition it is necessary to examine it regularly and also keep the mine roadway properly maintained.

In running a conveyor:

(1) Remove any spilled coal from around the drive and take-up heads, and on the floor under the cradles and guards.

(2) Make sure the carrying and return idlers rotate freely and have their shafts securely held in place by lock nuts.

(3) See that the belt moves without tending to run to one side, the joint fastenings hold reliably and the belt surface is not damaged.

(4) Check the belt cleaners for proper contact with the belt. Whenever necessary, pull the rubber wipers further outward.

(5) Examine all bolted joints to see whether they have worked loose; tighten them if necessary. Keep watch on oil seals so as to prevent dirt and dust from getting into the gear box and bearings.

(6) Never allow gummings to gather at the belt, cradles and guards. When gummings gather on the cradles or troughing, the belt will wear rapidly because it travels over a layer of coal instead of over clean idlers. Clogging by the gummings also tends to overload the drive motor.

(7) See that all the idlers are always in place. A missing roller means that the belt will sag, strike against the edges of the cradles, and suffer premature and swift wear.

(8) Never lag the drive drums with wood or rubber; it is difficult to make both drums of the same diameter, and the belt will slip on the drums and undergo accelerated wear.

(9) Never apply rosin, sand, etc., to the drive drum; this will only result in quick deterioration of the belt.

(10) Avoid chuting oversize lumps of coal (greater than 300 to 400 mm in size) onto the belt, or the belt will sag too much between the idlers. Also see to it that the coal is loaded onto the belt from a low height.

(11) Keep constant watch on the electrical equipment of the conveyor and clean the motor enclosure regularly of coal dust. Gummings and dust on the motor hinder its cooling, which may lead to a dangerous rises in temperature and outage of the motor.

The supply cable connections should be regularly inspected and the power contacts in the magnetic starters timely cleaned of deposits and oxide.

(12) Pay constant and close attention to the earth connections; the safety of the operators is fully dependent upon their continuity and solid contacting.

(13) For trouble-free operation of the conveyor regularly lubricate the following parts:

   (a) drive unit gear box;
   (b) drive drum gearing and bearings;
   (c) take-up and delivery pulley bearings;
   (d) motor bearings;
   (e) take-up worm gearing;
   (f) carrying and return idler bearings.

The gear box is lubricated by filling it with a mixture of 50 per cent cylinder oil and 50 per cent motor transmission oil to a level not higher than the check plug opening and at least 9 cm above the bottom of the casing. A mixture of 15 per cent of solid oil and 85 per cent of motor transmission oil will also serve this purpose. To prevent oil leakage a tight joint should be maintained between the gear box casing and its cover, and the felt seals watched for deterioration.

The oil is poured through an opening in top of the top cover. Old or dirty oil can be drained through the opening provided in the bottom of the casing. The oil should be changed at least every month.

After the oil has been drained, the gear reduction casing, gearing and bearings should be washed with kerosene, the worn seals replaced and the casing filled with fresh oil.

The gears of the drive drums should be greased at intervals of one month, stuffing solid oil between the gear teeth through the opening in the upper part of the casing. Every time the grease is renewed the gears should be washed with kerosene.

The drive drum bearings are generally lubricated with solid oil and should have the grease replaced every two months. The bearings should be first washed in kerosene and then stuffed with solid oil.

The worm gearing of the belt tensioning devices should have gear or solidol grease applied directly to the wormwheel teeth.

The grease in the idler ball-bearings must be changed at least every month, after they have been washed in petrol. The grease in ball bearings should be changed on the surface rather than under-ground.

Mounting of Belt Conveyors

Before a conveyor is set up, make sure that the working in which it is to be operated is straight. This can be checked by stretching a cord from end to end of the working. If the latter is straight, instal-lation is begun with the drive head. It must be set so that the axes of the drive pulleys are at right angles with the direction of belt travel. The motor bedframe is then placed on either the left or right side of the belt, and the motor and gear box are set up on it. With the drive head in place and anchored, the cradles are set up, beginning from the drive head. The cradles next to the drive head are mounted on wooden sleepers of progressively decreasing height. The subsequent cradles are placed on the floor along the entire length of the conveyor. Care must be taken to set the return idlers at a right angle to the di-rection of belt travel. The idlers must be spaced on 1400 mm cen-tres. Where the conveyor is to serve as a stationary installation, the cradles should be mounted on wooden boards laid lengthwise to the conveyor axis, and fixed to them by spikes. The return idlers are at-tached to the cradles in the surface shop before the cradles are sent underground. After the cradles have been set in place, the carrying idlers are mounted.

Enough room must be left in front of the take-up head towards the end of the heading for belt tensioning. When the two heads and the intermediate cradles are all in place, the belt, which is moved about rolled in coils, may be run out. To do so, each coil is put on a steel bar which is then placed on two special horses just forward of the take-up head, and the belt end is passed round the drum, over the guide roller, through all the cradles and over the return idlers towards

the drive head. As each section ends, a new section is attached to it. When the bottom strand has been run out the full length of the conveyor as far as the drive drum, the bottom-belt guards are put in.

The free end is now passed round the drive drums and the motor jogged to pull the belt over the carrying idlers towards the take-up head. When both ends of the belt meet, they are jointed to make the belt endless and the tensioning is begun. To do this, wooden struts must be set up back of the take-up head and the tensioning ropes tied to them. Rotation of the handles on the tensioning device will move the head towards the struts as the ropes wind up on the drums. The tensioning is continued until the sag in the belt between the idlers is fully taken up. The following step is to start the motor and move the take-up head and delivery pulley so as to make the belt run on the drums. It may happen that the belt tends to run to one side at the take-up or drive head, or on the carrying or return idlers.

When the belt tends to run to one side of the take-up head, the tension must be increased on the same side. Should the belt tend to move to one side on the cradles, the cradles are out of line, and the misalignment must be eliminated, while checking all the idlers for freedom of rotation. The belt may also run to one side when the idlers on one side are somewhat lower than those on the other side. If the belt runs to one side at the drive head, the carrying idlers incorporated in the head must be adjusted for proper position, or the head itself realigned, if it is out of line with the conveyor axis.

After the belt has been adjusted and runs correctly, all the idlers should be firmly secured in their final position, the belt cleaners brought in contact with the belt across its entire width, and the conveyor given a trial run under load. After the belt is put in operation, its tension must be adjusted from time to time, as it will stretch and the slack must be taken up.

Prevention of Accidents During Operation of Conveyors

To prevent accidents, the fundamental rules listed below must be observed at all times:

(1) Conveyors must never be used for man-riding.

(2) A conveyor should never be made to carry concentrated loads (machine parts, large lumps of rock, etc.).

(3) Never attempt to replace idlers, screw down grease cups, replace belt cleaners, remove the guard at the drive head coupling, or clean the drums of gummings, while the conveyor is running.

(4) Electrical equipment shall always be de-energized before it is repaired.

Belt Conveyor Troubles, Their Causes and Remedies

Trouble	Cause	Remedy
Belt runs off drive drums	Drive head out of line with conveyor axis	Eliminate misalignment, set drums at right angles to conveyor axis
	Drive drum shafts out of parallel with each other	Set shafts in parallel
	Misaligned delivery pulley shaft	Eliminate misalignment; make sure shafts of delivery and drive pulleys are parallel
Belt runs off take-up pulley	Belt misaligned at take-up end	Check take-up head for proper installation, adjust tension on both sides of belt
	Faulty joint in belt	Locate faulty joint and re-make it
Belt runs to one side on top and bottom idlers in the middle of conveyor	Cradles are out of line with each other and set askew	Straighten conveyor, level off bad cradles with blocks of wood
	Troughing idlers are misaligned or inclined against direction of belt travel	Adjust idlers for proper position
	Belt joined out of square	Rejoint belt
	Offset load on belt	Eliminate offset
Belt sides rub against drive head frame; head is mounted correctly	Slack belt	Take-up belt slack with screw take-up of delivery pulley
Bottom belt strand rubs against cross channel where belt leaves drive head	Missing bottom guide roller	Put in missing bottom guide roller
Belt slips on drive drums	Slack belt	Take-up belt slack
	Gummings on drums, idlers or belt	Remove gummings
	Lump of coal or stone, or piece of prop under belt	Remove any foreign objects
	Belt fasteners have become unbent or pivot pins at belt joints have become loose or protrude and catch at section sides or make belt seize	Check belt joints and eliminate faults
	Wet belt	Protect belt and drive head from dripping moisture
Belt wears badly	Causes are the same as when belt slips, and also:	
	Gumming on idlers	Remove gummings

Trouble	Cause	Remedy
	Missing idler Layer of coal dust on drum Dirty belt	Put in missing idler Keep drums clean See that belt cleaners operate properly
Abnormal knock-ing in gear box	Worn gears and bearings Gears do not mesh properly due to improper assembly The oil has become dirty	Replace worn gears and bearings Check gear box for any faults in assembly Replace the oil
Gear box and its bearings become overheated	Oil dirty; lack of lubricant Damaged bearings	Wash out casing and bear-ings; refill casing with fresh oil Replace
Oil leakage from gear casing	Missing gasket under oil drain plug Bad sealing of joint be-tween casing and cover Worn felt seals	Place gasket Check joint throughout its length for adequate sealing Tighten bolts which secure cover to casing Replace
Drive drum bear-ings run too hot	Grease dirty or absent Faulty bearing or bearings Misaligned pulley	Replace grease Replace Realign
Gummings clog take-up head at bottom of pulley and at guide roller Fine coal sticks to pulley	Rubber wiper does not make contact with belt sur-face	Adjust wiper for contact with belt
Motor hums when started and fails to come up to speed	Burned contacts in starter Blown fuse starter box One conductor in supply cable burned through or dis-connected from motor ter-minal Conveyor clogged by gum-mings and small coal; lumps of coal and stone, wooden debris between belt and a drum	Clean contacts Replace Restore connection, or re-place cable if faulty Remove gummings and small coal; extract jammed objects which cause stoppage

5-3. Belt-type Loaders

Loaders of the belt type are employed in heading work so as to fully utilize the productivity of a stone loading machine or a header and reduce the time lost in the exchange of loaded and empty cars. This type of loader can alternately load coal or stone into empty cars run under its jib one after another. In some schemes of mechanization these loaders are used in wide-heading entry drivage to drop the stone onto a shaking conveyor or onto the apron of a scraper haulage unit for delivery to a packing point or waste floor.

The ПT-1 (PT-1) belt loader is a track-riding truck. The turntable of the loader accommodates a short belt conveyor complete with its gear box and a 2.7-kW drive motor. Its design stone-loading capacity is 40 cubic metres per hour and it can deliver stone at heights of 0.5 to 1.45 mm above floor level. The length of the loader (without the conveyor) is 1755 mm, the conveyor being from 2300 to 3000 mm long. The maximum height of the delivery end is 1740 mm. The width and weight of the unit are, respectively, 2975 mm and 1300 kg.

The УПЛ-1c (UPL-1s) suspended-type belt loader is designed to accommodate a train of 15 one-ton mine cars under its boom. Its conveyor is assembled of a series of sections flexible in a vertical plane owing to hinge joints. The conveyor is suspended from the roof supports by its hangers. As the heading is advanced, the loader is pulled up by the stone loading machine or header. The total length of the loader, including the conveyor, is 33.23 metres. It is powered by a 5.5 kW motor, has a weight of 2876 kg and a design loading capacity of 40 cubic metres per hour.

B. MINE WINCHES AND FACE LOADING POINT EQUIPMENT

Rope haulage may be of three main types: namely, main or direct rope employing one rope, main-and-tail rope employing two ropes, and endless rope. The main field of application of rope haulage is on inclined roadways.

5-4. Endless-rope Haulage Winches

In endless-rope haulage the pulling rope strand hauls loaded cars along the "full" track, while the return strand transports the empties over the "empty" track. By reason of this, an endless-rope haulage requires a double track roadway.

Both the empty and loaded cars are attached to and detached from the rope "under way" with various types of grips, either in the form of an eccentric lever-movement or "drum" gripping device. Sometimes a wedge type grip may be employed.

Endless-rope haulage systems are operated by winches incorporating friction pulleys. To maintain adequate tension in the rope and prevent it from slipping on the drum, a tensioning arrangement with a balance weight and tensioning pulley is provided.
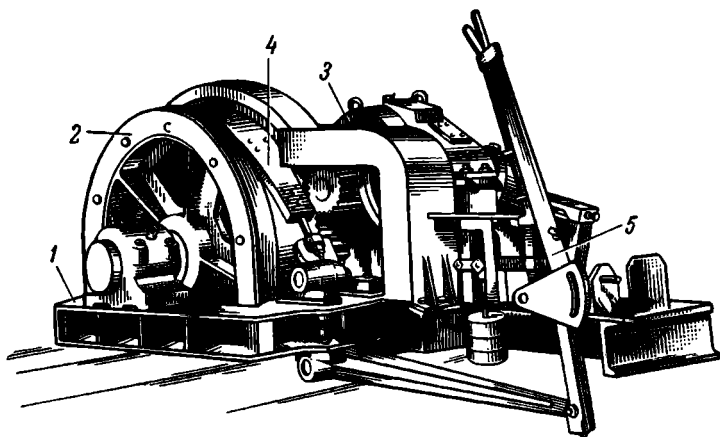
Fig. 54. ОЛ 1200/60 м (OL 1200/60 m) haulage winch

Since this type of haulage is employed on inclined roadways, runaway car safety dogs or catches must be used. Such catches are mounted between rails at definite distances from one another. When a car travels downhill at normal speed, the catch dogs are depressed by the car axle, but the catch hooks which rise at the same time, drop out of the way soon enough to permit the car to pass further downhill. However, should a car become detached from the rope and gain excessive speed, the hooks catch at the car axle because they will be unable to drop back fast enough.

Type ОЛ (OL) haulage winches for endless-rope transport (Fig. 54) comprise a bed frame *1*, friction pulley *2*, gear reduction *3*, emergency band brake *4* and operator's station *5*. In addition to the emergency brake, these winches are fitted with a haulage control brake.

The rope makes two and one-half turns round the friction pulley to have a large arc of contact—a factor upon which the tractive force on the rope is dependent.

5-5. Main-and-tail-rope Haulage Winches

Main-and-tail-rope haulage is very seldom used on horizontal road-ways, being mainly employed for car and skip haulage on inclined roadways.

The most common driving device for single-rope (direct) and two-rope (main and tail) haulage is a single-drum haulage winch, and less frequently a double-drum winch with a main and tail rope.
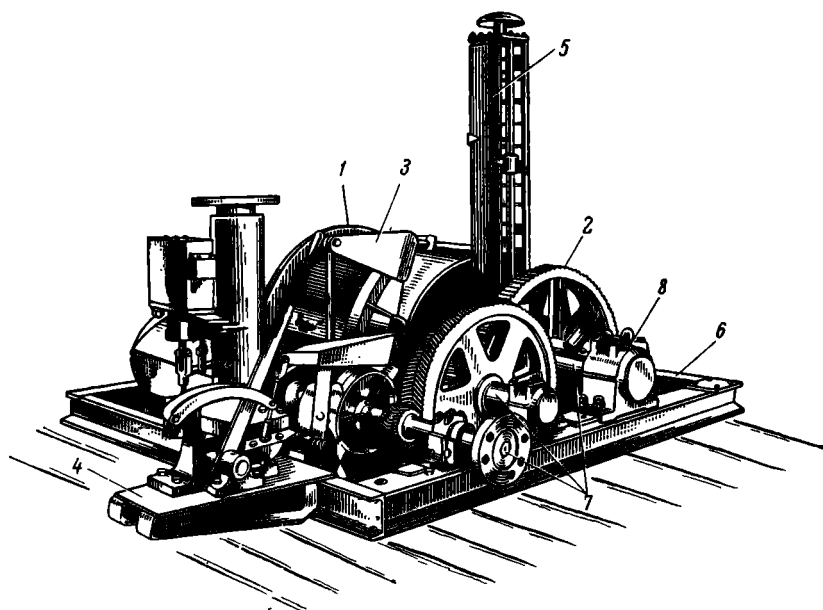
Fig. 55. БЛ 1200/1030у (BL 1200/1030u) haulage winch

The БЛ(BL) single-drum winch (Fig. 55) consists of a rope drum *1*, reduction gearing *2*, emergency shoe-type brake *3*, control station *4*, depth indicator *5*, bedframe *6*, shafts *7*, bearings *8*, and a haulage control brake.

The depth indicator shows the position of a car or skip on the incline. As the name implies, the emergency brake is used in an emergency stoppage of the winch whenever a trouble occurs. Operative stopping of the winch and braking during lowering of the load is performed with the haulage control brake.

In a double-drum winch one drum is keyed to the shaft, the other not. Whenever it becomes necessary to alter the length of the ropes,

the drums, usually bolted to each other, are detached and the neces-
sary drum turned to shorten or lengthen the rope.

To decrease wear of the sleepers, rollers are installed between the
track rails. At curves the rope runs round guide rollers.

5-6. Car Spotting Winches

Car spotting or gathering winches are employed to handle cars at
loading points. They pull the rope at low speed.

Car-gathering winches may be of the single or double drum type.
Their drive is generally an electric or an air motor.

The МЭЛ-4.5 **(MEL-4.5) winch** (Fig. 56) is built with a bedframe
1, electric motor *2*, and a drum *3* with a built-in speed reducer using
two internal-tooth gear trains and a planetary-friction clutch trans-
mission. A band brake *4* is used to brake the rope drum, while anoth-
er band brake *5* serves to start and stop the drum with the aid of
the planetary-friction clutch transmission.
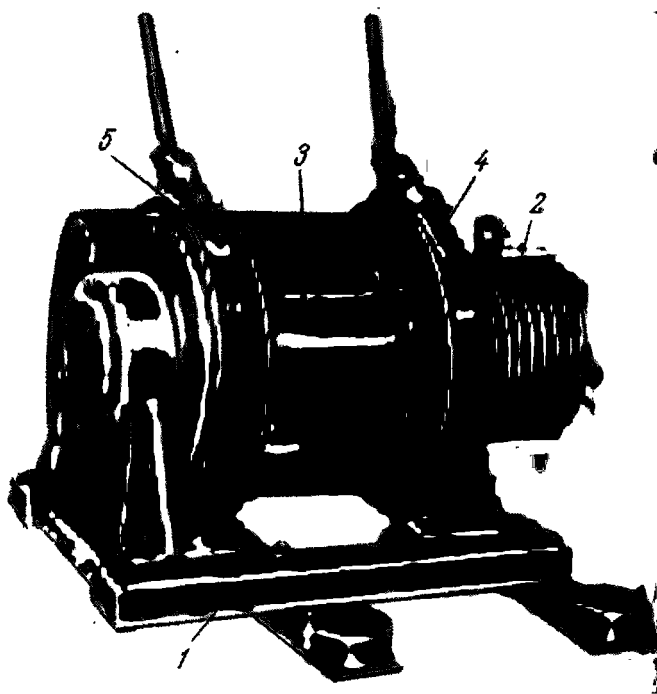
Fig. 56. МЭЛ-4.5 (MEL-4.5) car gathering winch

The **МЭЛ-11.4 (MEL-11.4) winch** differs from the MEL-4.5 winch only in size and tractive force.

The **МПЛБ-7 (MPLB-7) winch** is of single-drum design and is powered by an air motor.

MK-3 and MK-4 winches are double-drum units. The internal faces of the drums, and both ends of the helical gear are provided with cam jaw surfaces. The lateral thrust developed under load by the helical gear reduction, causes the follower gear wheel to shift along the shaft to engage with one of the drums. The engaged drum is thereby rotated. When the drive motor is reversed, the other drum is driven.

ЛМЭ-4.2 (LME-4.2) and ЛМЭ-11.4 (LME-11.4) winches are single-drum units similar in design to the MK-3 winch with one of the drums replaced by a braking plate with seats in its surface for engaging with the cams.

It should be noted that car-gathering winches may be fitted for manual or remote control.

5-7. Loading-point Equipment

Loading points where coal is loaded into cars from face conveyors are generally equipped with car or gathering winches.

On pit-bottom sidings and at the surface of a mine, cars are pushed through a limited distance by car pushers. They may be of either the chain, rope or piston type.

In a chain pusher the chain is fitted with dog catches to engage the car at its bumper or axle.

A rope pusher has the dog attached to its rope which moves alternately to and fro.

A piston pusher consists of one or two actuating cylinders with piston rods to which dog catches are attached. By feeding oil under pressure (in the case of an electrohydraulic system), or compressed air (in the case of a pneumatic system) into the cylinders, the dogs are traversed forward and returned by the piston or pusher rods. As the dogs move forward, they engage the car to accomplish the required movement; on a return stroke they drop and pass clear of the car.

Where mine cars are unloaded into bunkers, tipplers are used to turn them over.

C. ELECTRIC AND PNEUMATIC DRILLS, HAMMER DRILLS, DRIFTING-BORING MACHINES AND DRILLING RIGS

In stoping and heading work a great deal of shot hole drilling and shot firing has to be done. Shot holes are drilled with electric, pneumatic or hydraulic drills; pneumatic hammer drills are also employed.

5-8. Hand-held Electric Drills

Several types of hand-held electric drills are produced in the U.S.S.R. for mining work. Typical of them are the drills discussed below.

The **СЭР-19Д (SER-19D) electric drill** consists of an aluminium body, an electric motor, a fan seated on the motor shaft, reduction
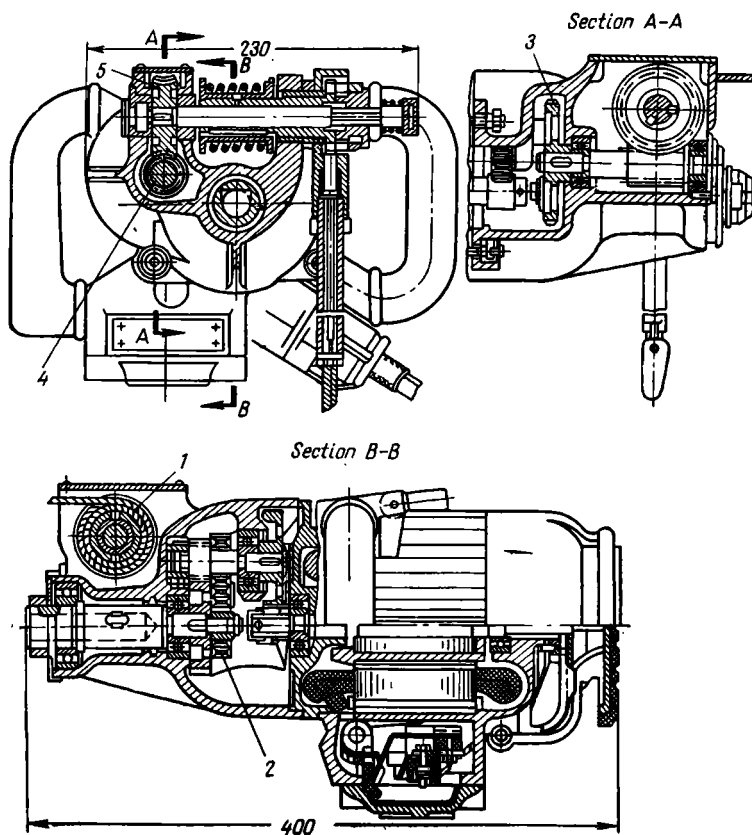
Fig. 57. ЭРП-6 (ERP-6) hand-held electric drill

gearing, a spindle and chuck for attachment of the drilling tool, a switch and a cable coupler. The axial fan is housed under a hood, rubber-lined on the inside and serves to cool the drill motor. Change gears permit the reduction gear to drive the chuck spindle at 338 or

690 rpm. This drill is controlled remotely through a five-core cable circuit fed from an АП(AP) starter unit (see Chapter VIII).

The СЭР-19 (SER-19) drill differs from the above drill in that it is fitted with a hand-operated direct supply switch. The СЭР-20 (SER-20) and СЭР-20Д (SER-20D) drills have motors of greater power rating.

ЭР-4 (ER-4) and **ЭР-5 (ER-5)** electric drills incorporate a switch for hand control which connects the motor to the supply when the drill handle is gripped.

The **ЭРП-6 (ERP-6)** electric drill (Fig. 57) is of somewhat different design. It has a feed arrangement whereby the drilling tool can be forced against the face when driving a hole. Drill feed is accomplished by a small-diameter wire rope, one end of which winds up on the drill drum, with the other end anchored to a prop. As the spindle of the electric drill rotates it also revolves drum *1* to take up the wire rope and thereby smoothly feed the drill into the face. The reduction gearing in this drill consists of a pair of spur gears *2* and *3*, and a worm *4* and wormwheel *5*.

The **ЭРП-20 (ERP-20)** electric drill is an improved version of the ЭРП-6 (ERP-6) drill. Its weight has been reduced to 20 kg.

The **СВЧ-2 (SVCh-2)** electric drill operates on a frequency of 150 c/s from a flameproof ВПЧ-150 (VPCh-150) frequency converter. Owing to this, the drill attains high power at a low weight.

5-9. Pneumatic Hand-held Drills

In headings with no power supply available shot holes can be drilled with pneumatic hand drills such as the СПР-11(SPR-11) shown in Fig. 58 and the СПРП-15-1 (SPRP-15-1).

These pneumatic drills essentially consist of a rotary air motor *1* with a regulator *2*, planetary gearing *3*, and a spindle *4*. The rotary air motor has a cylindrical body (stator) within which a slotted rotor with radial sliding vanes rotates. The rotor revolves with a certain amount of eccentricity relative to the centre line of the body. The СПРП-15-1 (SPRP-15-1) pneumatic drill differs in that it uses an air leg.

5-10. Hydraulic Hand-held Drills

The **СРГ-3 (SRG-3)** hydraulic drill is employed in the hydraulic mining of coal and incorporates a hydraulic motor (turbine) operated by water at the pressure used for supply of monitors. Its hydraulic motor develops 2.5 hp at a water pressure of 30 atm (gauge) and a water flow rate of 5 litres per minute. The spindle runs at 650 rpm and the drill weighs 12 kg.

Section AA-BB-CC-DD

Section EE

Fig. 58. СПР-11 (SPR-11) hand-held pneumatic drill

5-11. Operation and Maintenance of Hand-held Electric Drills

Before any new drill, or one just received after repair, may be handed over for use at the face, it should be checked as follows:

(1) The insulation resistance between the phase windings and between the windings and the drill body must be measured. It must be not less than 150,000 ohms.

(2) The drill switch must be checked for proper functioning.

(3) The drill must be checked for correct operation by running it idle for 30 minutes; the temperature rise of the body must not exceed 25°C.

Prevention and Remedy of Troubles in Operation of Electric Drills

Trouble	Cause	Remedy
Motor hums abnormally, spindle fails to turn	Motor receives supply only from two phases because of break in third phase of motor, connector or cable core	Locate break in phase and reestablish circuit. If break is in motor, turn in drill to workshop for repair
Spindle runs in wrong direction	Improper connection of phases	Switch off power to starter and interchange any two of the drill terminal connections
Motor comes up to speed with difficulty when started, hums and quickly heats	Rotor rubs against stator because of improper assembly or worn bearings	Turn in drill to workshop for repair
Drill body is at a potential	One of the cable cores is in contact with drill body where insulation is damaged. Stator winding has a fault to earth	Restore core insulation to prevent any possible contact with body. If winding is defective, send drill to repair workshop
Excessive temperature of drill body	Blunt drill bit, insufficient lubrication, low supply voltage, bad contact in switch, rotor rubs against the stator	Locate and eliminate trouble. If drill continues to run hot, send it to repair workshop
Motor starts but switches off	Broken holding spring in drill switch	Replace
Knocking in gearing	Foreign object in gearing or broken part in gearing	Open gear casing to locate the cause; remedy trouble or send drill to repair workshop

(4) The cooling fan must be checked to see that it runs properly and that its vanes do not strike or rub against the casing.

(5) The bearings and the reduction gear must be checked for lubrication; when grease is added to the bearings, care shall be taken not to allow any dirt to get into the bearings.

The driller, before beginning work, must examine his drill and flexible cable and see that the earthing connections are properly made. No work should be begun if the driller finds that a cable core has become bared or that the cable has been spliced without vulcanizing the joint. After the defects have been remedied, and everything is in order, the driller may insert the plug into the gate-end starter box and run the drill idle.

The driller must not begin work until a test has indicated that the methane content is low enough and that the face has adequate ventilation.

If the drill is not to run hot, the air intake openings must always be kept clean. If not, the insulation of the stator winding may be heated dangerously.

The bearings require a consistent grease such as grade УТВ (1-13) [UTV(1-13)] for their lubrication. The reduction gearing is lubricated with solid oil, care being taken not to fill the free space in the gear casing more than half full. This grease must always be kept clean and carefully handled to prevent dirt, cuttings and metallic particles from getting into the gearing where they may cause rapid wear or breakage to the gears.

5-12. Post-mounted Electric and Pneumatic Drills

Post-mounted electric and pneumatic drills are placed either on posts set up directly in a heading, or on the manipulators of stone loading machines. Drills of this type can drive holes in rock of medium hardness.

In Soviet mining practice the ЭБК-2А (EBK-2A) and ЭБК-2м (EBK-2m) post-mounted electric drills have found wide application. They are nearly identical in design.

Drills of more recent design are the ЭСГП-4 (ESGP-4) and ЭСГЛ-1 (ESGL-1) post-mounted electric drills which incorporate hydraulic feed of the drill stem, and the ПСГП-1 (PSGP-1) pneumatic drill.

The ЭБК-2 (EBK-2) post-mounted electric drill. This drill is built with an aluminium body having two chambers, one for the drum-type starter, the other for the used oil.

Fig. 59a gives the gearing diagram of the ЭБК-2м (EBK-2m) drill. Referring to it, the shaft *1* carries the motor rotor and a pinion *2* meshing with gear *3* seated on a common shaft with gear *4*. From gear *4* motion is transmitted through gears *5* and *8* to gear *6*. The latter
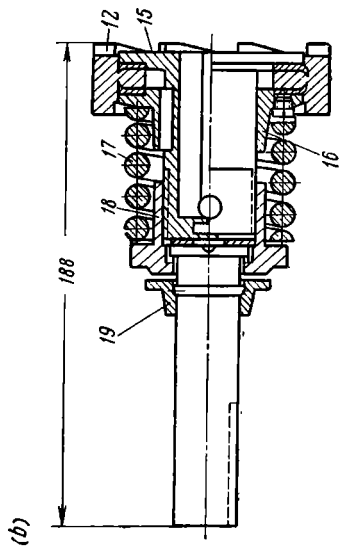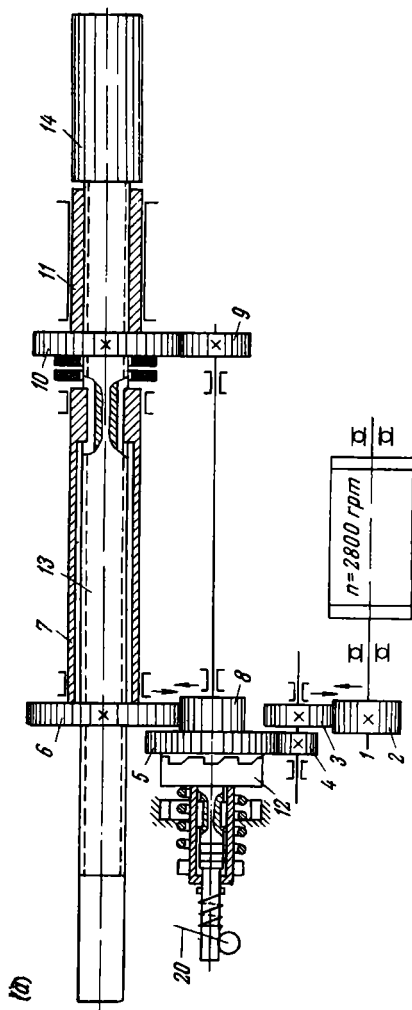
Fig. 59. ЭБК-2м (EBK-2m) electric drill:

a—gearing diagram; *b*—clutch

is bolted to a guide sleeve 7. On the inside, sleeve 7 has two opposing keys which fit into longitudinal slots in a drive screw 13. The end of this screw carries a drill-rod chuck 14. Gear 5 has a toothed rim for engagement with a clutch 12. Two recesses are machined in clutch 12 (Fig. 59b), one to receive sleeve 15, the other to receive sleeve 16. The sleeve 16 can move over the sleeve 15 on a key. A spring 17 is fitted between the nut 18 of the switching arrangement and sleeve 16 to hold the latter tight against clutch 12. The pressure between them is adjusted with a nut 18.

Thus, when gear 5 rotates (see Fig. 59a), sleeve 15 also rotates (when gear 5 and clutch 12 are engaged) to transmit motion through the key to the shaft of feed gear 9.

Feed gear 9 meshes with gear 10 fixed on nut 11 through which drive screw 13 is passed. The drill rod is fed into the face as follows.

Motion from the motor rotor is transmitted via gears 2, 3, 4, 5, 6 and 8, to guide sleeve 7, and through its keys to drive screw 13. If the clutch has been thrown in, gears 9 and 10 and nut 11 will rotate in synchronism, but at a greater speed than does the drive screw. This results in rotation of both the drive screw and the nut. Since the nut revolves at a greater speed, the screw will move forward and out of the nut (feed toward the face). When the clutch is withdrawn from engagement, the drive-screw nut will have to rotate together with the drive screw due to the friction between them. Consequently, there will be no relative motion between them, and the screw will not advance (the drill bit will turn in a bore hole without feed-in). When the clutch is braked (with the ring 19), the gears 9 and 10, and, consequently, the nut 11 come to a standstill. This makes the screw, now revolving in a stationary nut, travel backwards. The clutch is thrown in or out with a handle 20.

The frame of the motor in this post-mounted drill is not fan-cooled; the winding has waterproof insulation, but is not oilproof.

Motor control is provided by a drum starter placed in a separate chamber. The starter handle has the shape of a wing nut.

Lubrication. The drill is lubricated with cylinder oil every shift, the oil being introduced with an oil can through ball valves 1, 2, 3 and 4 (Fig. 60). Once a week the drill gear chamber is filled with a mixture of grade УC-3 (US-3) grease with an addition of 10 to 15 per cent of cylinder oil. This lubricant is added through openings 5 and 6. The bearings require greasing at bi-monthly intervals.

The following rules must be observed in operating a post-mounted drill:

(1) The temperature rise of the drill body must never exceed 40°C; when this occurs, it indicates that the rate of feed does not correspond to the hardness of the rock being drilled, that the drill bit has become dull, and that the drill itself may be at fault.

(2) To avoid wear to the drill mechanism, the clutch should never be thrown in or out with the drill running.

(3) Scheduled maintenance must be done as prescribed and the drill must be regularly lubricated.

(4) A special appliance should be used to guide the drill stem when a hole is bored; the drill should never be guided with the hand.

(5) No drill should be operated until its body has been earthed

(6) The post supporting the drill should have its clamping screw tightened from time to time to maintain the post rigidly fixed.
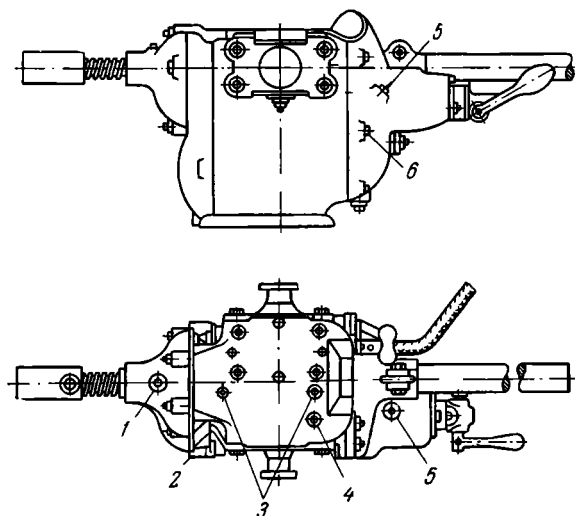
Fig. 60. Lubrication points of ЭБК-2м (EBK-2m) electric drill

The **ЭБК-5 (EBK-5)** post-mounted electric drill is an improved version of the ЭБК-2 (EBK-2) drill and incorporates a two-speed electric motor in which the number of pole pairs can be changed by a switch. This makes it possible to run the electric motor at a speed near 3000 rpm or 1500 rpm and to halve the speed during hole drilling. The ЭБК-5 (EBK-5) drill also includes a device for adjusting the axial force by changing the spring pressure in the friction clutch without any need for dismantling the gear casing.

The ЭБК-5 (EBK-5) electric drill can be locally or remotely controlled.

Where power is supplied at 660 volts in the mine, the ЭБК-5 (EBK-5) electric drill employs a 127-volt motor fed from an АПК (APK) starting box (see Chapter VIII).

The ЭСГП-4 **(ESGP-4) post-mounted electric drill** has hydraulic instead of mechanical feed, accomplished with an oil gear pump built into the reduction gear. Rotary motion from the drill motor rotor is transmitted to the oil pump to deliver oil under pressure to the hydraulic cylinder in which the piston has a stroke of 900 mm. Feed pressure on the drilling tool can be adjusted over a range from 150 to 1000 kg and set by a valve incorporated in the hydraulic system.

The ПСГП-1 **(PSGP-1) post-mounted pneumatic drill** is of the same design as the ЭСГП-4 (ESGP-4) drill, but instead of electric drive it has a rotary air motor which develops 5.5 hp at 3.6 atm (gauge) pressure.

The ЭСКЛ-1 **(ESKL-1) light-weight post-mounted electric drill** is fitted with hydraulic feed. It weighs less and has a lower drilling capacity than the ЭСГП-4 (ESGP-4) drill.

5-13. Hammer Drills

Shot holes in medium and hard rock are driven with a hammer or percussive type drill. According to use and weight, hammer drills are classed into four types: light-weight (up to 20 kg), medium-weight (up to 30 kg); post-mounted units (up to 100 kg), and telescopic drills (weighing from 25 to 50 kg and designed to drill shot holes straight up). Telescopic hammer drills use a column support and are extended by compressed air.

Electric or pneumatic drills operate on the rotary principle. The hammer drills break down the coal and stone in driving a hole by percussive-rotary action.

In a pneumatic hammer drill, compressed air alternately enters the spaces on either side of the piston, making the striker travel back and forth in the cylinder and strike at the drill shank at the end of a work stroke. At the end of each stroke the air distributing system automatically changes the direction of the compressed air to return the striker and, at the same time, turn the drill rod through an angle with the aid of a rod turning mechanism. Each successive blow of the drill bit in the bore hole thereby occurs at a new spot in the hole bottom.

As a bore hole is sunk, coal or stone "cuttings" or dust gather and must be removed to permit the drill to work normally. This can be done with compressed air or water.

The latter can be accomplished either with central or side flushing.

In hammer drills employing central flushing the wash water is fed through a metal tube in the drill to the hole or channel in the drilling rod. When the drill is designed for side flushing, the water is directed down the drilling rod through a sleeve.

Where wet drilling is impossible to accomplish, the cuttings are removed dry. For this, the drill is fitted with a dust exhausting tube communicating at one end with a hole in the drilling rod and at the other end with an ejector located in the head of the hammer drill. The dust can be sucked out of the hole by the drill ejector and discharged into a dust catcher where it can settle down.

Since the stone dust produced by drilling is dangerous to breathe because it leads to diseases of the lungs, measures must always be taken to control it by wet drilling or installation of a dust exhauster system.

When shot holes have to be bored in any possible direction, rising holes included, it is possible to use a light-weight ПР-13Л (PR-13L) hand hammer drill.

For ease of handling hand type hammer drills, various kinds of air legs, columns or manipulators such as the МБИ-5y (MBI-5u) or МБМ-2 (MBM-2) units are mounted on stone loading machines.

The **ПР-30Л (PR-30L), ПР-30ЛВ (PR-30LV) hammer drills,** using central and side washing respectively, are designed to drill downward and horizontal blast holes without the aid or support columns or posts.

Recent practice has shown a trend towards the use of "fast-striking" or "high-frequency" hammer drills capable of up to 3000 to 3500 blows per minute, whereas the ordinary hammer drill operates at a rate less than 2000 blows per minute. With fast-striking hammer drills, especially in conjunction with appropriate pneumatic columns, drillers can attain considerably higher outputs per shift.

The **ПРЛ-24Л (PRL-24L) fast-striking hammer drill** (Fig. 61) is intended for work in horizontal and inclined workings. It operates on a pneumatic column fitted with built-in air controls, so that air to the column is fed from the same hose as the drill.

The **ПРП-24Л (PRP-24L) fast-striking hammer drill** has a built-in pneumatic column. Incorporated in its body is a feed control device and a button for quick release of the air pressure in the pneumatic column when the drill rod jams or when the drill rod is returned to its starting position. This drill is attached to the pneumatic column by means of a quick-locking joint arrangement.

The **ПР-18Л (PR-18L) fast-striking hammer drill** is intended for drilling shot holes in rock of medium hardness where narrow headings have to be driven.

The **КЦМ-4 (KTsM-4) post-mounted hammer drill** can drive shot holes as deep as 10 metres in hard rock and is mounted on a support column or carriage. This drill has a starting valve with four positions: (1) full flow, with compressed air passed through the full cross section of the intake ports and a partial blow-out taking place at the same time; (2) partial flow, at which the intake ports are only partially
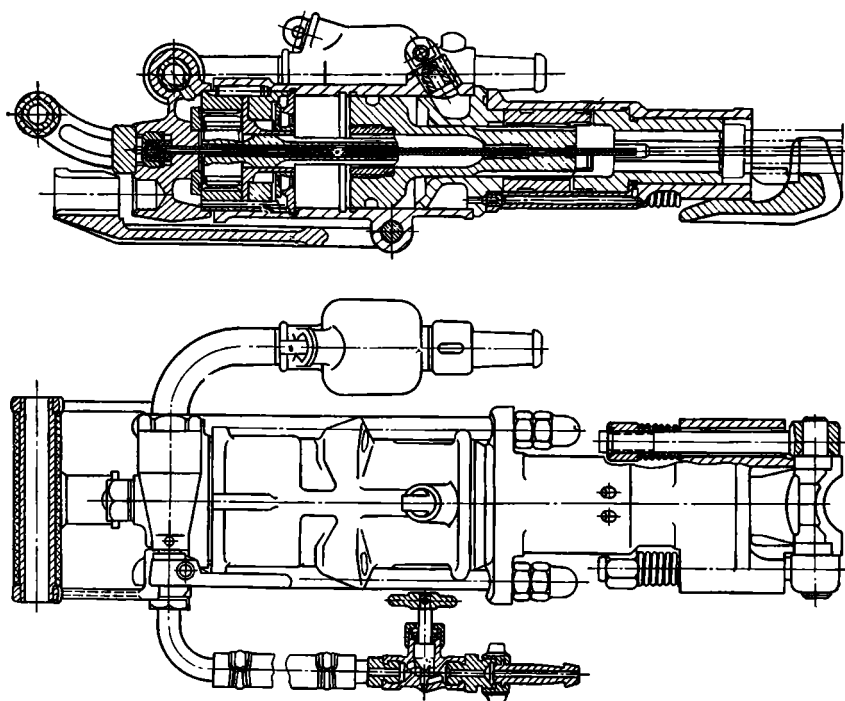
Fig. 61. ПРЛ-24Л (PRL-24L) hammer drill

also be mounted on a rig truck or carriage. The БК-2 (BK-2) truck-mounted drill rig has two booms or jibs, raised or lowered by a worm screw mechanism. When operating at a face, the truck is rigidly anchored in position by its screw-type thrust columns.

The ТП-4 (TP-4) telescopic hammer drill is adapted for drilling uprising blast holes with pneumatic feed. Its control valve has four positions: (1) stop, at which compressed air is shut off from the drill; (2) short-time interruption, during which the pneumatic feed has its piston moved forward and the drill is stopped without being lowered; (3) start feed; (4) drill, at which the drill is operative. The ТП-4 (TP-4) drill is designed for wet drilling.

The ПТ-45П **(PT-45P) hammer drill** differs from the ТП-4 (TP-4) drill in that it has an ejector for removal of the cuttings.

The ПТ-30 **(PT-30) hammer drill** weighs one-third less than the one above and has been designed to bore shot holes to shallower depths.

Efficient operation of hammer drills demands that work and maintenance rules be observed, namely:

1. Compressed air must be delivered at a pressure not less than 5 atm (gauge). The rate of boring drops on lower pressures.

2. The compressed air fed to the drill must be kept clean and free from moisture. The air mains should always be fitted with moisture traps.

3. Before a hose can be connected to the drill it should first be blown out.

4. The drill rod shank must fit snugly in the drill chuck with its end truly perpendicular to the chuck axis, and the length of the shank should exactly correspond to the depth of the chuck seat.

5. Hammer drills should be lubricated at least twice during a shift. Grade Л(L) or heavier machine oil mixed with 25 per cent of kerosene may be used for this purpose.

6. The form of the drill bit must correspond to the rock being drilled. To drill rock of medium and high hardness, chisel or carr bits are needed. When soft or friable rock is drilled, multipoint (cross) bits are best suited for the job.

7. The water pressure in wet drilling should be at least one atmosphere below that of the compressed air.

8. When a hammer drill is hand-held during drilling, it has to be kept in line with the bore hole axis.

9. A hammer drill should never be allowed to operate idle, not only because of waste of compressed air, but also because it may lead to a breakdown of the drill.

10. Drilling should never be carried out with dulled drill steels. It results in a sharply reduced rate of boring.

11. Hammer drills require oiling either by hand or by automatic oilers placed in the air supply line or incorporated in the drill itself.

12. If a hammer drill must be lubricated by hand, it should be oiled at least two or three times a shift.

13. Every time a detachable bit is screwed onto the drilling stem, its threaded end should first be lightly greased or oiled.

The efficiency of a hammer drill depends to a great extent on the conditions of the drill steel. The latter comprises a bit, and a hollow drill stem with a shank, and shank collar or shoulder. The bit is usually detachable and is manufactured from a high-alloy steel while the stem is made from a cheaper grade of steel.

Drill bits are essentially of the chisel (carr), or cross type. Carr type bits tipped with wedge-like inserts are used in solid rock with

hardnesses up to 10 on the Protodyakonov scale, bits of the same type with rectangular inserts are used in rock with a hardness of 20 on the Protodyakonov scale. Friable rock of any hardness is drilled with cross type bits tipped with wedge-form inserts.

The insert plates are generally cemented carbides (tungsten carbide, etc.) and are brazed in the slots of bits with red copper.

Dulled bits should be sharpened on a bit grinder in the drill dressing shop. The shanks of the drill stems must also be dressed in this shop.

For re-use, drill steels should be reconditioned as follows. After being cleaned of dirt, drill rods are heated in an oil or electric furnace to 1100 C and then taken to a drill rod shaping machine where their shanks or heads are forged to proper shape and size in dies. After the shaping operation, the drill rods are annealed by heating to 740°-850°C and slowly cooling in sand, and then hardening their shanks by reheating them to 850°C and quenching in oil.

When properly shaped and dressed, a drill steel shank should satisfy the following requirements:

(1) The shank end face to which the striker blows are applied must be smooth and at right angles to the drill rod axis;

(2) The shank axis must coincide with the drill rod axis;

(3) The shank dimensions must comply exactly with the dimensions prescribed by the relevant drawing;

(4) The hole through the shank and drill rod must be clean and free from obstructions.

Elimination and Prevention of Troubles in Hammer Drills

Trouble	Cause	Remedy
Boring rate of drill is low, drill rod turns with difficulty	Dulled cutting edges on drill steel, too much stone or coal dust in hole	Replace drill steel or its detachable bit, blow or flush out cuttings
Seizure of stem-turning bushes	Worn turning bushes; bad alignment of shank bush; poor lubrication	Replace worn bushes; oil drill
Excessive compressed air consumption; low boring rate	Hammer drill parts are worn; loose joints	Replace worn parts; see that threaded joints are made up properly
Seizure of striker	Expanded striker end Poor lubrication	Replace striker Lubricate drill

5-14. Holing Machines and Drilling Rigs

Holing machines and drilling rigs serve to drive holes in coal. Holing has to be done in pitching seams to cut rise headings (rise entries, raise chutes, stall roads, etc.). Raise work is usually carried out by first driving a hole and then widening it to the necessary cross section with air picks or by shot-firing.

Holing machines serve to make upward holes in coal 300 to 390 mm in diameter and to enlarge them to diameters ranging from 500 to 900 mm.

The CБM-3y (SBM-3u) holing machine (Fig. 62) carries a boring head comprising a spiral boring tool *1* of 107-mm diameter, a forward-cutting reamer and a rearward-cutting reamer. The boring tool is driven and fed by electric or air motor *2* through gear box *3*. The boring tool is connected to the drive spindle and rods by a column coupler *5*. The drill rods, each 0.6 m long and joined together one at a time, make up a drill column. When each consecutive rod must be added, the column is held in the raised position by a clamp *7* supported by a pair of parallel posts *6*. The drive spindle can then be lowered, and the next drill rod inserted between the spindle and the drill column. To prevent deviation of the bore hole from a straight line, guide collars are fitted in the drill column every six to eight drill rods.

After the boring tool has completed the 390-mm bore hole, the rearward-cutting reamer is fitted to expand the hole to 850-mm diameter by return from top to bottom.

For transportability in the underground workings, the CБM-3y (SBM-3u) machine is truck-mounted and its frame *4* travels on track wheels. During boring the machine is anchored by means of screw-extended posts the lower ends of which are set in seats provided in the frame, while their upper ends are jacked against nearby roof supports in the working.

When boring is carried out at inclinations up to 45 degrees, use is made of auger-type drill rods so as to remove coal cuttings from the hole.

The MБC-2 (MBS-2) holing machine is designed for work from gate roads. It is smaller in size than the CБM-3y (SBM-3u) machine.

The drill column in the MБC-2 (MBS-2) machine is fed into the hole by two hydraulic jacks secured on extending guides in the machine. A pump unit driven by a 0.9 kW motor supplies the working fluid needed for operation of the jacks.

The БPM-2 (BRM-2) holing machine differs in design in that the drill column does not rotate and primarily serves to deliver compressed air to the boring-head air motor at the hole face. The drill column also serves to feed the boring head. Boring head feed-in is accomplished

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2293

8

2

4

2222

2100

940

460

7

1

6

5

3
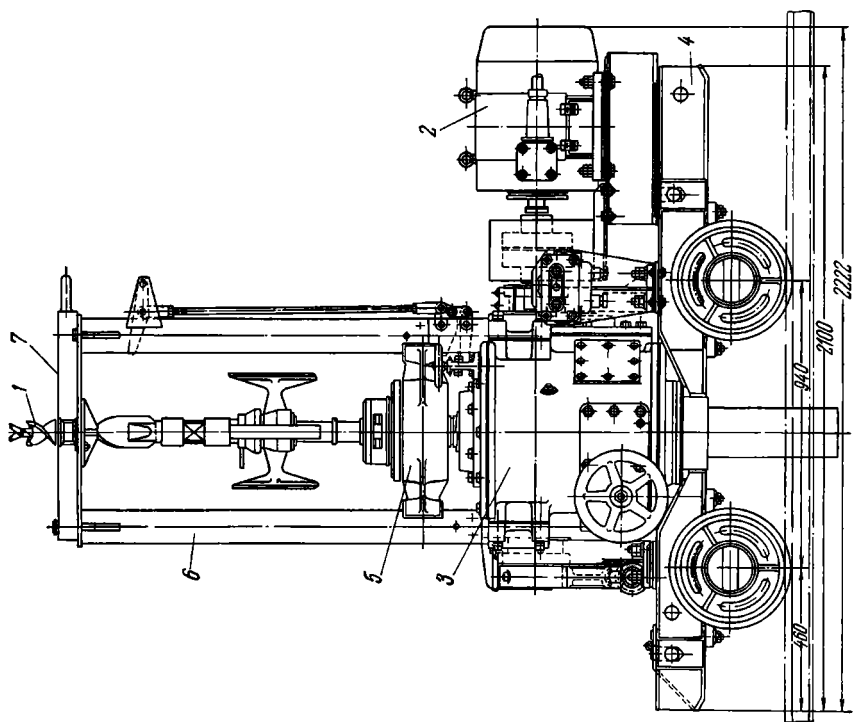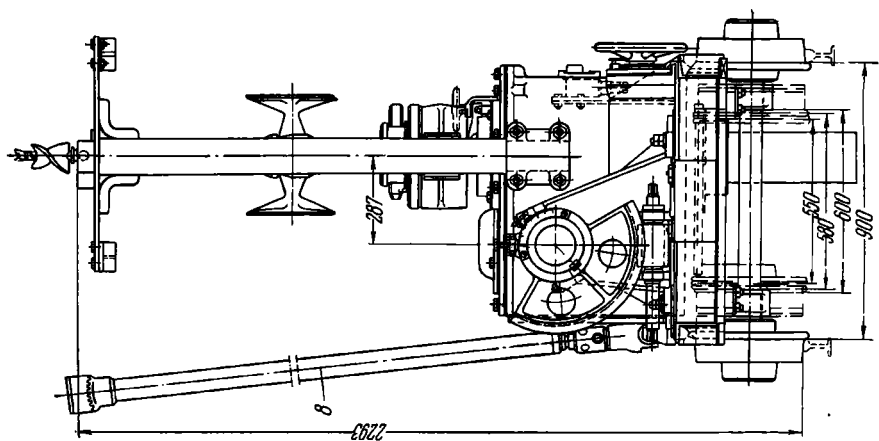
by hydraulic jacks. The hydraulic system pump in this machine is driven by a 3-hp air motor.

The light-weight ЛБС-4 (LBS-4) holing rig is an improved model of the ЛБС-2 (LBS-2) rig. It comprises a body and reducing gear case mounted on a frame, an electric or air motor, and a drill column fitted with a boring tool. The latter consists of a boring bit and a reamer, both of which are attached to the end of the drill column, the latter consisting of a series of rods coupled to the rig spindle. To prevent bending, the drill rods are fitted with guide collars at intervals of six to eight rods. The spindle in this rig not only rotates the drill column, but also feeds its forward. The column is extended in the same way as in the СБМ-3у (SBM-3u) machine.

The rig body is carried by horizontal trunnions which are made fast to two uprights on the bedframe. The trunnions permit the body to be inclined to the required angle with a manually operated tilting mechanism. A check on the angle of boring is provided by an inclination gauge on the rig.

The БВУ (BVU) drilling rig is a unit used to drill bore holes of 92- and 180-mm diameter to depths from 8 to 15 metres at downward inclinations up to 5 degrees and upward inclinations up to 35 degrees.

The rig consists of a drilling tool, rotating mechanism, feed device, guide frame, and anchoring posts. A 5-hp air motor is used as the drive. The weight of the rig is 150 kg.

The БШ (BSh) drill rig is used for boring degassing holes in pitching seams in mines where the hazard of gas and coal outbursts is present. When boring upward at angles of from 50 to 90 degrees, the bore holes can be of 160-250- and 300-mm diameter; when boring downward, of 160 mm diameter. Holes from 120 to 150 metres deep can be bored with the rig. The drilling tool consists of a spiral and chisel type drill stem and auger-type drill rods. The rig has a drilling capacity of up to 50 metres per shift.

5-15. Air Picks

Air picks at the present time are mainly used for getting coal in thin pitching seams mined in steps.

The ОМСП-5 (OMSP-5) air pick consists, essentially, of an automatic starting arrangement and striking and air distribution mechanisms. The automatic starting arrangement operates in the following way. As long as the pick handle is not pressed, springs maintain it forced up, the intake-valve plate closes the compressed air intake passage, and the pick does not operate. When the pick handle is pressed, the intake (starting) valve is shifted to clear the passage, and air enters the pick.

The air is distributed by a slide valve and a set of radial ports and longitudinal passages in the valve chest and the pick barrel. As the compressed air flows through the pick during its operation, the distributing system in the pick makes the air first drive the striker piston forward until it hits the end face of the tool steel shank and rebounds. On the return stroke the air enters the barrel below the striker piston to drive it backwards to its initial position where the entire cycle is repeated all over again.
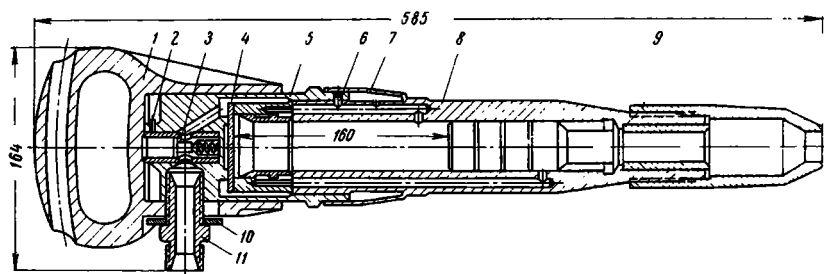
Fig. 63. MO-10 air pick:

1—handle; *2*—handle back-up spring; *3*—bushing; *4*—valve chest cover; *5*—intermediate unit and valve; *6*—stop screw; *7*—stop ring; *8*—barrel, valve chest and striker; *9*—tool retaining cap; *10*—bushing washer; *11*—nipple

At the present time the above pick is no longer manufactured, it has been replaced by the Model MO-10 air pick (Fig. 63) which uses a longer striker piston. The striker end enters the tool holder sleeve at the end of its work stroke, thereby producing an air cushion which protects the barrel from shocks. The tool steel is prevented from falling out of the pick by a retaining cap and shock absorbers.

The MO-8 air pick is lighter in weight and has a higher striking rate.

The MOM-10 air pick, which incorporates a sprayer, has been designed on the basis of the MO-10 pick. Its air and water inlet valves are so interlocked that no water will be able to enter the pick as long as the air remains shut off. The barrel of this pick is fitted with two nozzles by which water is directed against the coal face for wetting. A separate hose is run to the pick to feed water to it.

The Tomsk Electromechanical Machine Works is currently producing ОМП-8 (OMP-8), ОМП-9 (OMP-9) and ОМП-10 (OMP-10) air picks. They all have a simpler air distributing system as compared with the picks of the Leningrad Pnevmatika Works discussed above.

All the above air picks are designed to be operated at air pressures between 4 and 5 atm (gauge). Operation at pressures below rated value sharply reduces the productivity of air picks. Maintenance of air pressure is therefore a matter of prime importance.

Compressed air for air picks is supplied to the mine districts through air mains run from air-compressor installed on the surface. In order that air supply may be maintained at a constant pressure and free from water vapour and oil, the surface station is equipped with a main air receiver. Each main run to the mine districts is provided with an intermediate air receiver. It is the duty of the district electrical fitter to drain water and oil from these intermediate air receivers at regular intervals. He must also watch for air leaks in the mains. Air leakage does not only mean waste of electric power, but also reduces air pressure, and the air pick and hammer drill productivity will sharply decline.

The essential rules for efficient operation of air picks are:

(1) Hand-lubricated air picks must be oiled through the inlet opening 3 or 4 times a shift, with the air supply shut off and the air hose detached.

(2) Air picks oiled by automatic lubricating devices connected in the air line hose must have their respective devices filled with oil or the necessary lubricant once every shift.

(3) Every air pick should be given a preventive maintenance inspection and repair in the mine workshop every week.

(4) Every time an air hose is to be attached it must first be blown out with air.

(5) No air pick should be allowed to operate with worn parts as this leads to rapid rise in air consumption and sharp drop in productivity. The clearance between the barrel nose sleeve and the tool steel shank should not exceed 0.3 mm.

(6) Air lines should be connected and disconnected at the air mains or an air pick only after supply has been shut off at the main and air line valves. The length of the air hose line between an air pick and the air supply main should not exceed 12 metres.

(7) When a pick tool is inserted, it should be turned away from the operator to avoid an accident, should the pick receive air inadvertently.

Compressed air mains are installed in mine workings on brackets or suspended from hangers. In order to drain the water which condenses from the compressed air when the air cools down, the air mains must be laid with a slope of 1 in 200 to 1 in 300 in the direction of the air flow.

The individual lengths of pipe are joined together with a flange joint in which one of the flanges is welded to the pipe while the other is movable. To disconnect a joint, it is only necessary to loosen the nuts and turn the movable flange through 90 degrees. For air-tightness, the joints are made up with rubber-composition gaskets.

Air pick maintenance requires that picks should be given a quarterly routine repair and a medium repair every three months.

Elimination and Prevention of Troubles in Air Picks

Trouble	Cause	Remedy
Drop in rate of striking, some blows being too strong	Clogged inlet passages. Heavy air leakage through clearance between barrel nose sleeve and tool shank	Clean passages in barrel and air chest. Replace worn parts; clearance must not exceed 0.3 mm
Drop in rate of striking, some blows being too weak	Low air pressure in pick	Check air pressure, eliminate air leakage
	Constrictions in air hose due to separation of inner plies of rubber	Replace hose
	Dirty strainer screen in inlet bushing	Clean and wash screen
	Excessive wear of pick parts	Turn in pick for repair and replacement of worn parts
Seizure of striker piston or slide valve	Thickened lubricant	Wash in kerosene and refill with fresh lubricant
	Lack of oil	Lubricate pick
	Dirty parts	Turn in pick for servicing and repair
Pick continues to operate after handle is released	Inlet valve has seized, opposing spring missing	Grind valve surface smooth, put in opposing spring

D. COALCUTTERS AND CUTTER-LOADERS

5-16. Coalcutters

Coalcutters are designed to cut kerfs in the coal face preparatory to breaking down.

Soviet designers are continuously improving coalcutters. In particular, a great deal of attention is being devoted to make coalcutters fully automatic.

At the present time the coalcutters quantity-produced in the U.S.S.R. include the КМП-3 (KMP-3) and ПМГ-3 (PMG-3) and the more advanced Ural-33 and МВГ-2 (MVG-2) with hydraulic feed.

A coalcutter consists essentially of a motor, a haulage unit, cutting head, a bar or jib on which an endless cutting chain runs, and a gummer.

Rotation of the motor is transmitted to the haulage drum in the haulage unit and to the cutting head where the bar or jib takes off

power to run the cutting chain. The haulage drum carries the haulage rope, one end of which is clamped in the drum while the other end is tied to an anchor prop. As the drum rotates, it takes up the rope and pulls the machine towards the anchor prop. At the same time the cutting chain runs in the guide race of the bar, and the picks on the chain cut a kerf in the coal.

The КМП-2 (KMP-2) and КМП-3 (KMP-3) Coalcutters. The first of these machines is used in seams of bituminous coal and anthracites more than 0.5 m thick.

The gearing of the machine (Fig. 64). The left extension of the motor shaft, through gear *1*, rotates a shaft carrying gears *2* and *3*. This shaft is also coupled to the oil pump gears *7* and *8*.

The gear *3* transmits rotation to another shaft which mounts gears *4* and *9*. The gear *4* meshes with a flitting drive gear *5*, while the gear *9* engages the ring gear *10* on the friction clutch body. In the latter, the outer discs are fastened to the body (and, therefore, indirectly to the ring gear) while the inner discs engage with the hub on the shaft carrying worm *12*. Rotary motion is thereby transmitted to a wormwheel *13* through which the shaft-gear *14* can turn the rope drum because the drum ring gear *15* is in mesh with the gear *14*.

The gear *5* drives the internal-tooth half-coupling *6*. The other half-coupling *11* with external teeth is seated opposite on the shaft of the worm *12*.

If the half-coupling *11* is shifted to the right, it will engage the other half-coupling *6* and rotate the worm at flitting speed.

If the half-coupling *11* is shifted to the left, it will compress the clutch friction discs and the worm will begin to rotate at the speed of the ring gear *10*, or the maximum haulage or feed speed. The rope drum will rotate at this speed only if the friction clutch is kept compressed. If the friction clutch is alternately compressed and released, the machine will also travel and stop alternately.

The friction clutch *FC* (Fig. 65) is compressed when the tie rod *T* and the half-coupling *C* move to the left. The tie rod *T* is connected to a nut *N* the shoulder of which bears upon a lever *a*. The lever *a* will be moved by a push rod *A*, which carries a spring slipped over it. The end of the rod is fitted with a slide block *SB* set opposite the bevel face of the gear wheel *G* (gear *10* in Fig. 64).

If the slide block *SB* comes in contact with the face of the gear *G* at point *I*, where the gear has the least thickness, the gear, in rotating, will keep it continuously forced to the left. The push rod *A*, acting through its spring, will therefore press against the lever *a* and force the nut *N* and tie rod *T* to the left, thus maintaining the friction clutch *FC* compressed all the time, and the rope drum will have to rotate continuously. This corresponds to the maximum haulage speed of 0.86 m per min.

5*
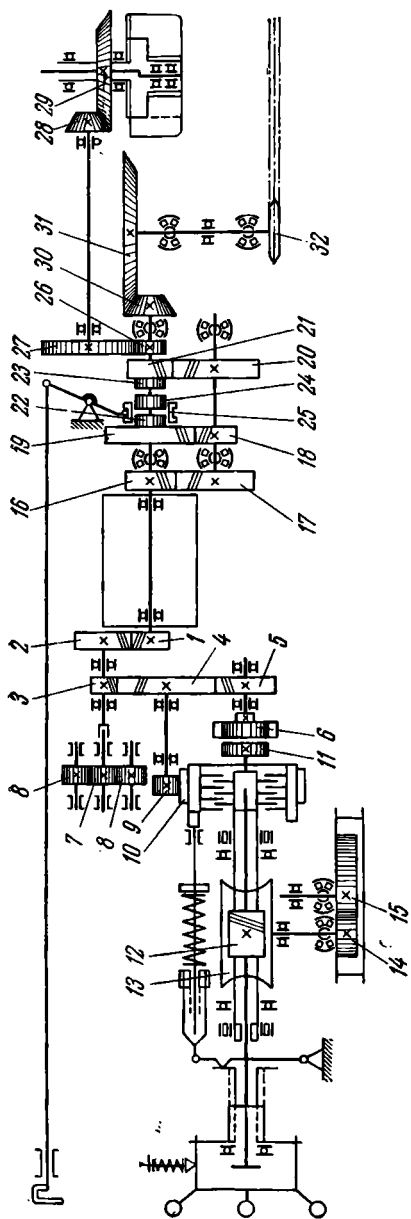
Fig. 64. Gearing diagram of the КМП-2 (KMP-2) coalcutter
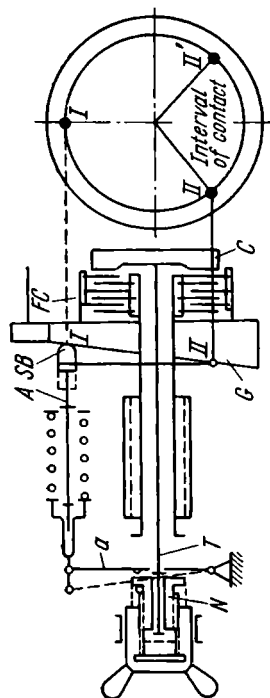
Fig. 65. Diagram of haulage-drive pulsator in КМП-2 (KMP-2) coalcutter

If the slide block *SB* is set to make contact with the face of gear *G* at point *II*, it will be forced to the left only as long as it takes the gear to turn from point *II* to point *II'*. During the remaining portion of each revolution, the slide block *SB* will be out of contact with the gear, and the friction clutch will be released. By reason of this, feed movement of the machine takes place once every revolution as the gear moves between points *II* and *II'*.

The above arrangement makes it possible to obtain any feed or haulage speed from zero to 0.86 m per min by properly setting the slide block with respect to the gear *G*.

The cutting chain is driven as follows (see Fig. 64). The motor pinion *16* transmits motion to the shaft on which gears *17*, *18* and *20* are seated.

The gears *18* and *20* mesh, respectively, with gears *19* and *21*, each free to run idle on the shaft. The same shaft carries a gear *24*. The gears *19* and *21* are also rigidly joined to gears *22* and *23*. An internal-tooth clutch *25* can couple the gear *24* to gear *22* or *23*. The shaft to which the gear *24* is keyed also carries two more gears *26* and *30* which will run at a speed, dependent on the position of the clutch *25*. The gear *30* serves to rotate the gear *31* and thereby drives the cutting-chain drive sprocket *32* keyed to the same shaft. The cutting chain therefore has two speeds. Gears *27*, *28* and *29* transmit motion to the gummer.

The Jib and Cutting Chain. The jib is a frame of bar steel to which steel plates are riveted. At the outboard end it carries a nose fitted with cover plates.

The cutting chain of the machine (Fig. 66) consists of a set of plate links *1* and pick boxes *2*. The links are joined to the pick boxes by pivot pins *3* held in place by spring lock rings *4*. The picks are fastened in the pick boxes by bolts *5*. The pick boxes may be of either the single or double pick type. The chain is laced to have seven or nine lines of cutting. The chain is kept taut by a tensioning screw.

The Cutting Head. The cutting head reduction gearing provides two speeds, the cutting speed (2.12 m/sec) and an idle speed (1.07 m/sec) for removal of gummings and replacement of picks. The reduction gearing (Fig. 67) is housed in a cast case closed with a cover. Each shaft and its gears is a separate unit, not connected to the top cover of the casing, and is therefore very easy to replace.

The gear *16* (the reference numbers in Fig. 67 are the same as in Fig. 64) is seated on the shaft of the motor and meshes with the gear *17*, both gears being placed in a separate oil bath. The gear *17* is keyed to the same shaft as the gears *18* and *20*. The second horizontal shaft supports the two free gears *19* and *21*, and also carries a keyed clutch. The clutch ring can be shifted by means of a fork connected to the tie rod. The length of the tie rod can be adjusted by means of

Section B-B

Section A-A

Pick lacing diagram for 9 lines of cutting

Chain travel direction
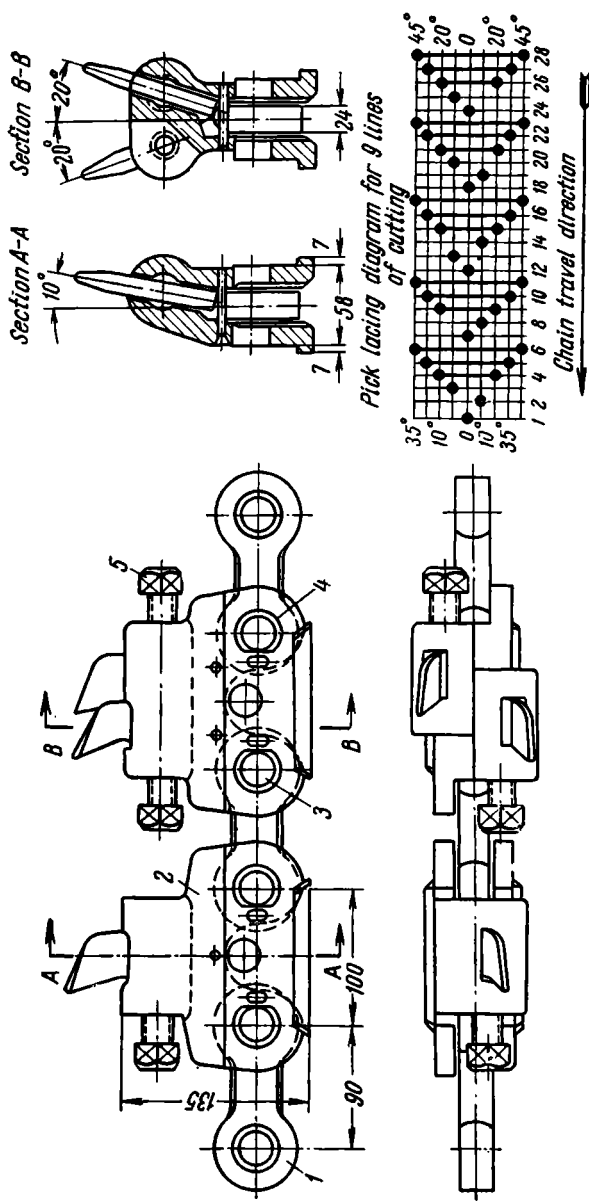
Fig. 66. Single-plate link, single-pivot cutting chain of КМП-2 (KMP-2) coalcutter
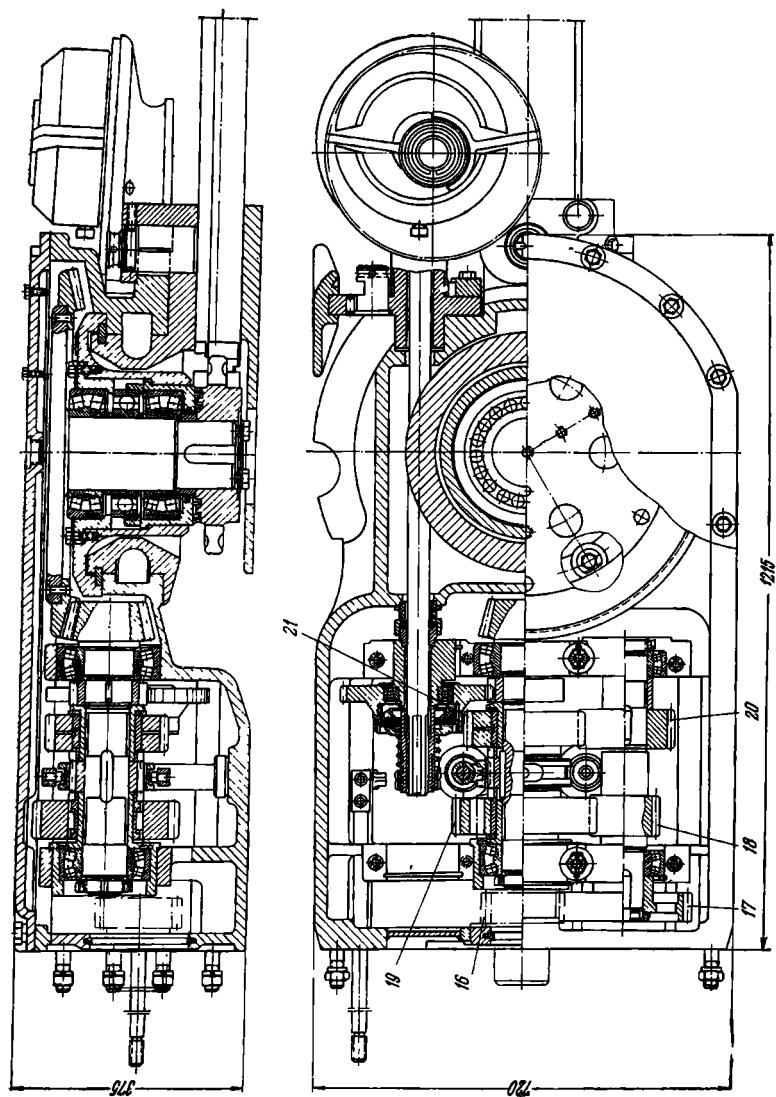
Fig. 67. Cutting-head gear box of КМП-2 (KMP-2) coalcutter

a sleeve. The tie rod terminates in a handle with a slot in the hub. The slot receives a pin fixed on the feed unit casing. This pin serves to set the handle of the cutting head in the following positions: the first outermost position, in which a cutting speed is obtained; the second outermost position, in which a servicing speed is obtained, and the middle position, in which the cutting chain is at a standstill.

The gear *30* (see Fig. 64) is made integral with the shaft and meshes with the bevel ring gear *31*. The hub to which the ring gear is attached also carries the chain driving sprocket keyed to it.

The gummer is a rotating drum with eccentric paddles which remove the gummings as they accumulate at the sprocket.

The gummer can only be set in its working position after the cutting jib has been swung round in readiness for a cut.

When the machine is to be flitted, the gummer must be set with its drum end upward, locked with the stop shaft, and the stop bolt lowered.

The haulage unit of the КМП-2 (KMP-2) coalcutter is housed in a casing jointed to the motor frame by means of studs (Fig. 68). A separate oil bath is provided in the casing for the motor pinion *1* and gear *2* which meshes with it (see Fig. 64). All the other reducing gears run in another oil bath.

The front side of the haulage unit carries the following controls: the handle of the cutting head, the "Start" and "Stop" buttons for remote control of the magnetic starter, a handwheel to control the haulage and flitting speeds of the machine, the controller handle, and the socket of the cable coupler.

In the latest КМП-2 (KMP-2) machines the conventional worm reduction in the haulage unit has been replaced by a globoid worm reduction capable of taking greater loads.

For proper assembly, the middle plane of the globoid worm gear ring must be aligned with the worm axis.

Adjustment of the axial position of the globoid worm relative to the wormwheel must be done in a machine shop with the use of a special template.

The electrical equipment of the coal cutting machine consists of an MA-191/10k electric motor, KPB-3006Б (KRV-3006B) controller, ШВ-9677 (ShV-9677) cable coupler and КУВ-6021A (KUV-6021A) push button stations.

The motor frame is a rectangular casting. The motor shaft bearings have seals. Any oil which may find its way into the motor along the shaft is whirled off by oil-throwing ridges into a trapping sump from which it can be removed. The fan mounted on the motor shaft sucks air through openings in the rotor and blows it through the ventilating ducts and the controller chamber. An interlock prevents the controller cover from being opened while the cable coupler is still plugged in.

Section A-A

Section B-B

View C
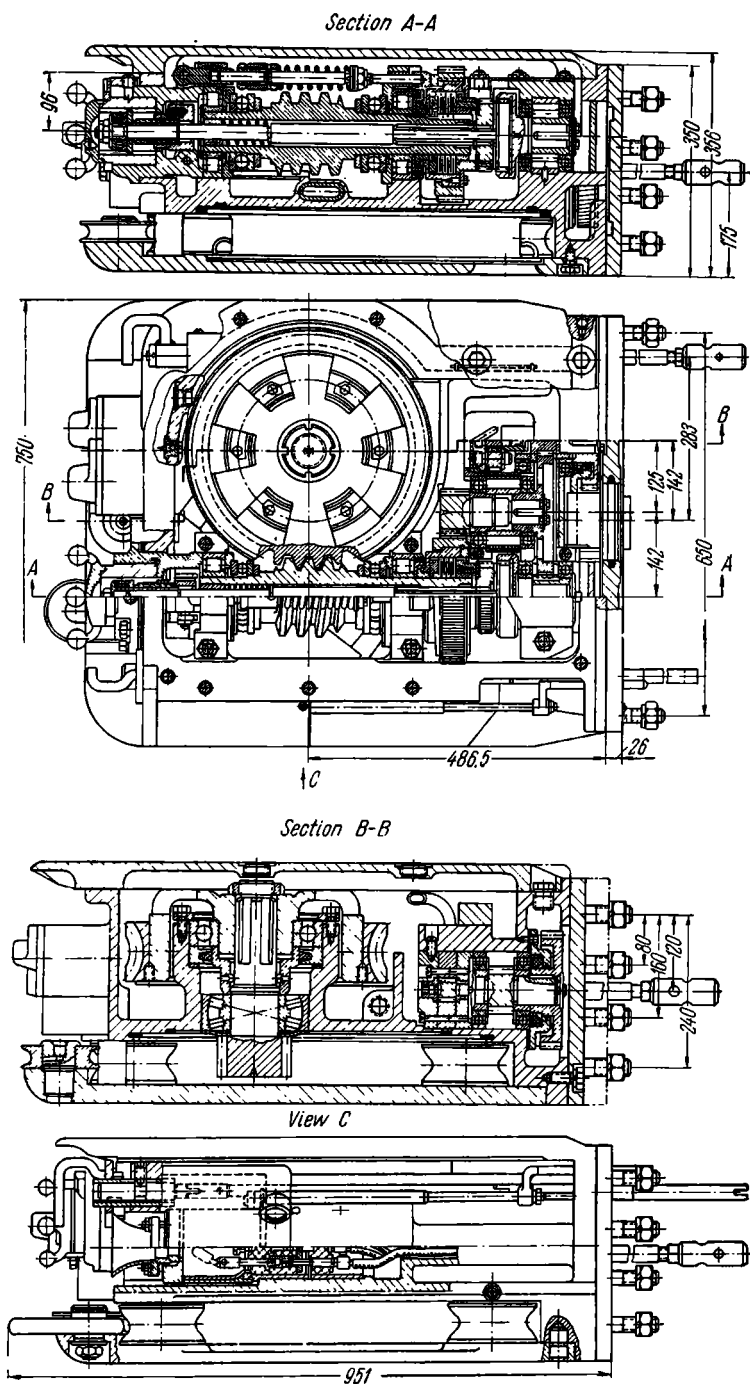
Fig. 68. Haulage unit of КМП-2 (KMP-2) coalcutter
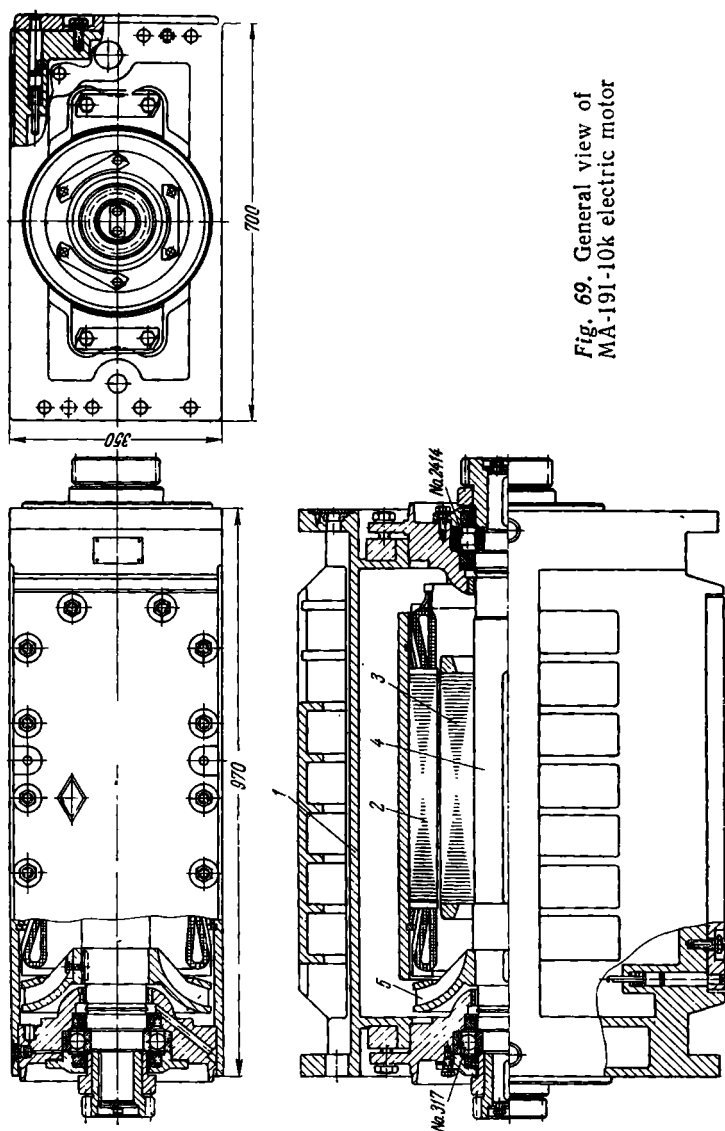
Fig. 69. General view of
MA-191-10k electric motor

Fig. 69 gives a general view of the MA-191/10k electric motor. The stator core *2* is assembled of electrical sheet steel laminations stacked and tightly locked inside the frame *1*. On the inner periphery the core has slots which receive the silicone-insulated stator winding.

The rotor *3* of the motor also has a core of electrical sheet steel laminations keyed to the motor shaft *4*. The slots in the core carry cast aluminium bars which form a squirrel-cage winding. The fan seated on the shaft *4* circulates air in the motor through the ducts provided in the rotor core, the air gap between the stator and rotor, the side air channel, and the controller chamber, closed by a cover. In its passage through the motor, the air transfers heat to the frame walls.

The antifriction bearings in which the rotor runs are housed in endshields. The endshields are fitted with felt seals and leather collars to prevent ingress of oil from the haulage unit and cutting head into the motor.

The endshields also have oil trapping sumps situated just opposite the oil-throwing ridges on the motor shaft. Any oil finding its way into the frame is thrown off into the sump from which it is drained through special passages. These passages must be cleaned daily with a piece of wire to maintain them unobstructed.

For better cooling, the motor frame is cast with fins to increase the surface.

The motor is started, stopped and reversed by the controller (Fig. 70) mounted in the chamber on the side of the motor.

The base *1* of the controller carries forward and reverse three-pole contactors *2* and *3*. These contactors are operated by a leverage system and a shaft *4*. Through electrical interlocks *5* with which shaft *4* is linked, the magnetic starter in the gate-end box is remotely operated. When any one of the two contactors (say, the forward contactor) is closed, the first to make are the main or power contacts, then interlock *5* closes to operate the gate-end magnetic starter. When the contactor is opened, the interlock breaks the control circuit before the main contacts are able to separate. Thus, when the motor is started or stopped, the power circuit is made and broken by the gate-end magnetic starter, and the main contacts of the controller remain unburned.

However, should a trouble develop in the remote control circuit, the motor can be locally started and stopped with the controller. To cope with such a duty, each pole in the contactors is fitted with an arc chute *6* for better arc extinction.

The trailing power cable is connected to the machine by means of a ШВ-9677 (ShV-9677) cable connector (Fig. 71). The socket *1* of the connector is fastened with a screw *2* in the casing of the haulage unit. The insulating block *3* of the socket has six pins *4* to establish circuit with the six cable cores (three power, two remote-control,

Fig. 70. General view of КРВ-3006Б (KRV-3006B) controller

ПОD

210

Section A-A

~ 400

46

22

121

View B

145D

23

46

40

80
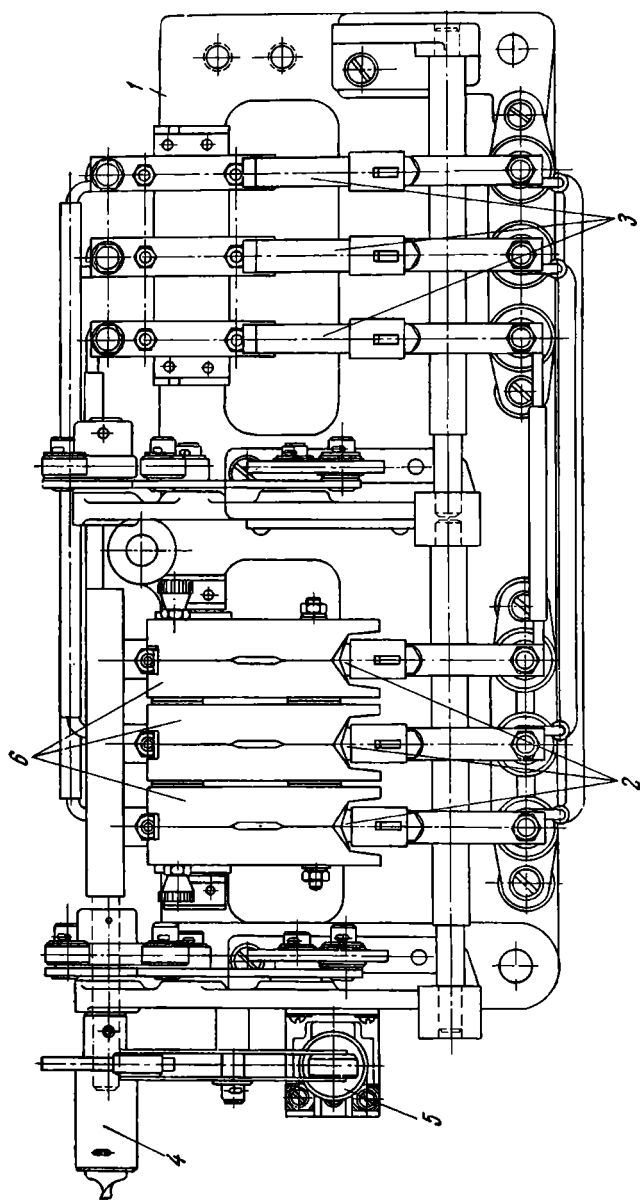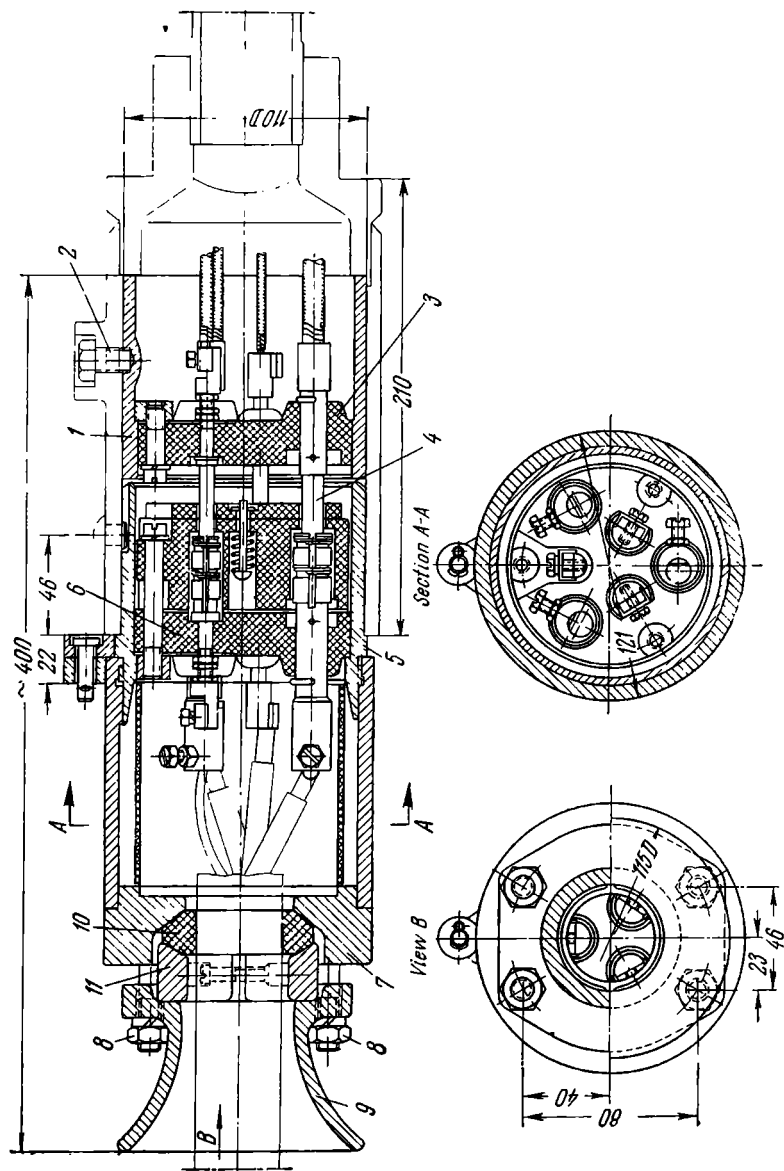
Fig. 71. ШВ-9677 (ShV-9677) plug-and-socket cable coupler

and one earth core). The plug shell *5* houses another insulating block *6* containing female contacts into which the pins slip when the coupler is joined. The plug shell *5* is attached to the body *7*, fitted with a gland endbell *9*. The latter is attached to the body by means of studs *8*. The cable is securely held in the coupler by a rubber ring *10* expanded by gland half-rings *11* when the endbell is made tight.
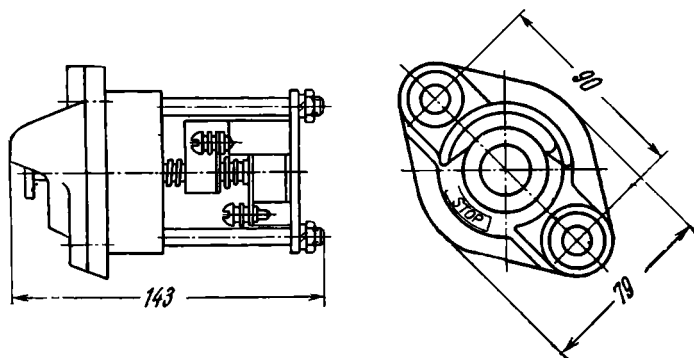
Fig. 72. КУВ-6021А (KUV-6021A) push button

The coupler is interlocked with the controller in the following way. A slotted ring is fitted on the controller handle. The coupler flange has a lug which can enter the slot only when the controller handle is in the "Off" position. As soon as the controller is switched on, the ring locks the coupler plug in and prevents it from being detached.

Two push buttons are fitted for remote power switching of the machine. Each consists of a steel case and an insulating block (Fig. 72). At the top and bottom the block carries a pair of fixed contacts between which a contact bridge connected to the button can be moved. When the push button is pressed, the contact bridge opens the upper pair of contacts and closes the lower pair of contacts.

When released, the push button is returned to its initial position by a spring.

Lubrication of the Machine. Liquid, semiliquid lubricants and grease are required by this coalcutter (Fig. 73). The grades used are: Industrial oil 30 (Cylinder oil L) and Cylinder oil 11, a semiliquid mixture of 50 per cent liquid oil and 50 per cent УС-3 (US-3) grease (solid oil T), and also УС-3 grease (solid oil T).

The antifriction bearings of the motor must be lubricated with УТВ (1-13) [UTV (1-13)] grease.

Table 4 lists the parts to be lubricated, the type of lubricant used and the intervals at which they must be lubricated.

View A

View B
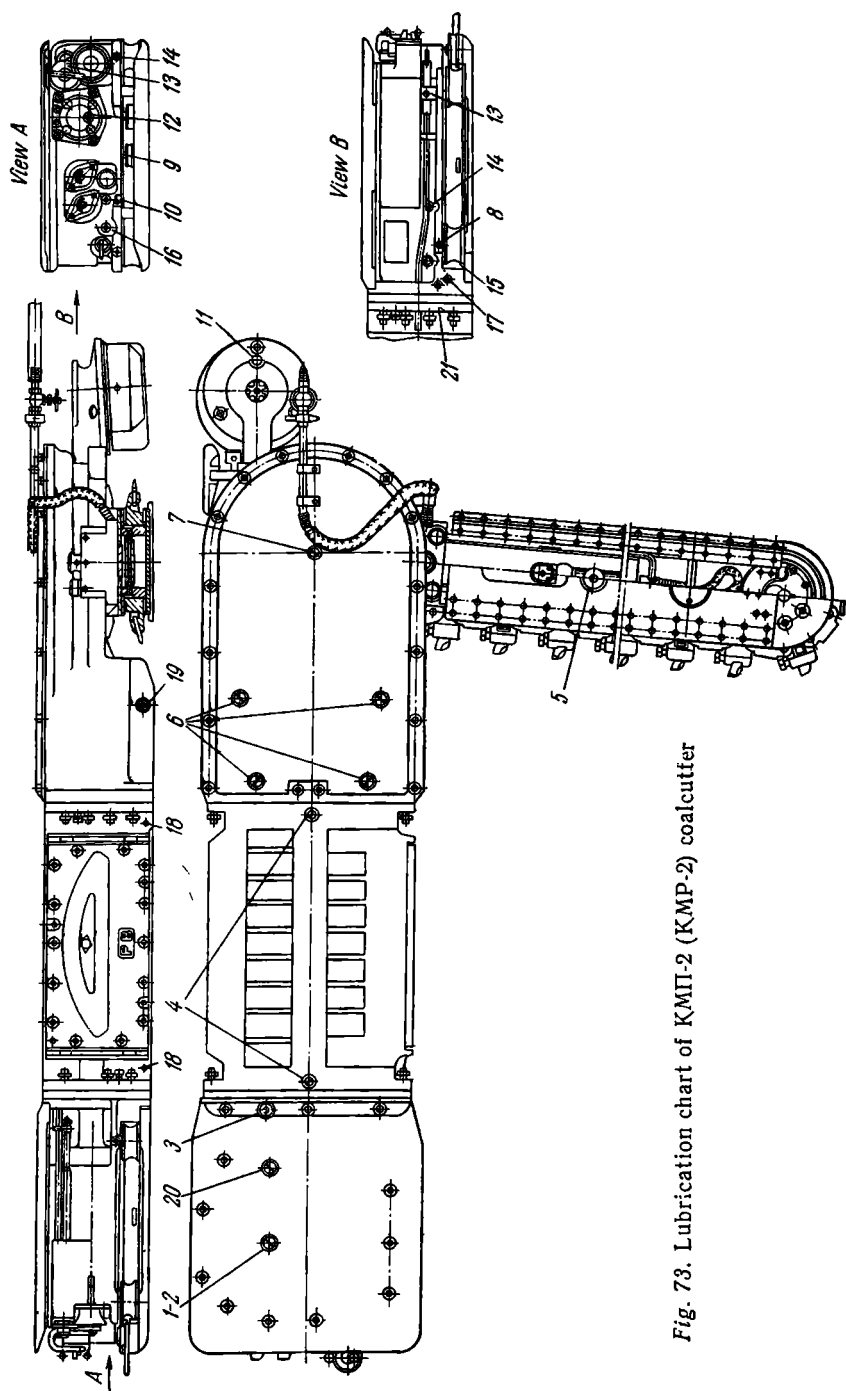
Fig. 73. Lubrication chart of КМП-2 (KMP-2) coalcutter

Table 4

Lubrication point (Fig. 73)	Operation	Lubricant	Periodicity
1	Lubricate worm and spur gearing of haulage unit	Liquid	At beginning of shift
2	Lubricate vertical shaft and wormwheel bearings	Grease	Every five days
3	Fill oil bath in first reduction gear chamber of haulage unit	Semiliquid	If oil is lacking as checked at point *21*
4	Replenish grease in motor bearing housings	Grease УТВ (1-13)[UTV (1-13)]	Bi-weekly
5	Oil chain tensioning parts	Liquid	At least once a month, after washing with kerosene
6	Lubricate cutting-head reduction gear	Semiliquid or liquid	At beginning of shift
7	Grease drive-sprocket vertical shaft bearings	Grease	Every five days
8	Drain oil from bath of worm reduction gear of haulage unit	—	Every month, with wash out of oil bath
9	Drain oil from pump and spur reduction gear in haulage unit	—	ditto
10	Drain oil from worm reduction gear when machine cuts from head end to gate	—	ditto
11	Drain oil bath of first reduction gear on haulage unit side	—	ditto
12	Drain oil getting into electric motor from bearing chambers	—	Every shift
13	Drain oil bath in cutting-head unit	—	Every month, with wash out of oil bath
14	Grease guide rollers	Grease	Every quarter
15	Grease central guide roller	Grease	At beginning of each shift
16	Grease rope drum bearings	Grease	Every 15 days

Table 4 (continued)

Lubrication point (Fig. 73)	Operation	Lubricant	Periodicity
17	Grease gummer reduction gearing	Grease	Every 10 days
18	Grease control handwheel thrust bearing	Grease	At beginning of each shift
19	Grease rubbing surfaces of control levers	Grease	At beginning of each shift after first wiping clean
20	Gauge oil level in haulage unit oil bath	—	At beginning of each shift
21	Check oil level in oil bath of first gear reduction	—	ditto
—	Oil link joints in cutting chain, chain proper, and jib	Used or residual oil	Before finally finishing work and at end of each shift before machine cools down

The Watering Device. Water is supplied to the machine by a hose line.

The upper saddle of the machine has two inlets, one for the right-hand and the other for the left-hand face cutting. The nipple-ended hose is screwed into the respective inlet hole, depending on the position of the coalcutter at the face. The other hole must be closed with a plug.

From the inlet, the water passes through one of the metal tubes laid inside the jib frame. The tube is joined to a rubber hose looped in the free space within the jib frame. The loop in this hose allows the cutting chain tension to be adjusted without breaking the water line connections.

At its delivery end, the rubber hose is attached to a bifurcated metal tube both ends of which are fitted with the spray nozzles. The latter direct the water jets into the kerf to keep the cuttings wet and thus suppress the dust. The pressure of the water jet at the nozzle should be between 4 and 5 atm.

The КМП-3 (KMP-3) coalcutter has been manufactured by the Kopeisk Engineering Works since 1958 instead of the КМП-2 (KMP-2) machine.

In the KMP-3 machine a friction clutch is incorporated in the haulage drive to protect the machine against overloads and breakage during tramming. The former jaw type clutch could not afford this protection. Owing to this friction clutch, the change from haulage

Fig. 74. Gearing diagram of КМП-3 (KMP-3) coalcutter

Cutting head
control handle

Flitting speed
control springs

Cutting and
flitting speed
control

Rope drum
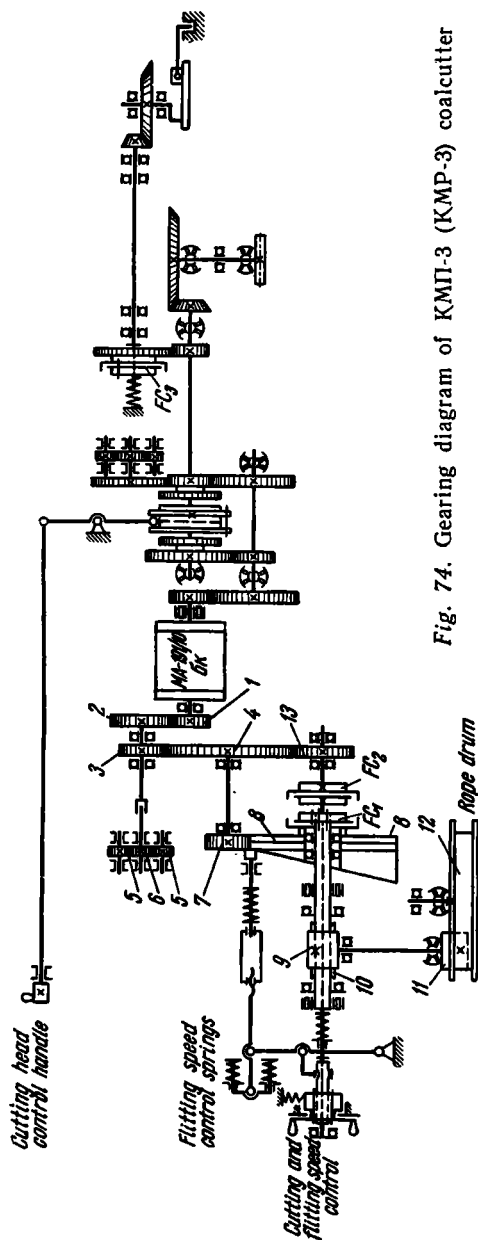
to tramming speed can be made on the run without stopping the motor.

The gearing diagram of the КМП-3 (KMP-3) machine is given in Fig. 74. When the machine operates at haulage speed, motion from the electric motor is transmitted to the haulage rope drum in exactly the same way as in the KMP-2 machine; i.e., through gears *1* and *2, 3* and *4, 7* and *8*, multidisc haulage-speed friction clutch FC_1, worm *9* and wormwheel *10*, and gears *11* and *12*.

At tramming or "flitting" speed, motion from the motor is transmitted to the drum through gears *1* and *2, 3* and *4* and *13*, multidisc flitting-speed friction clutch FC_2, worm *9* and wormwheel *10*, and gears *11* and *12*.

To switch over to tramming speed, the control handwheel is turned clockwise. This makes the buffer nut move to the left and shift the lever that pulls the pulsator control slide block out of contact with the bevel face of the gear *8*; at the same time the discs of the tramming speed clutch are compressed together.

Oil pump gears *6, 5, 5* receive motion from the shaft-gear *3*.

The haulage speed of the КМП-3 (KMP-3) machine can be varied over a range from zero to 1.42 m/min.

A new crank-driven gummer has been designed for use with the cutting head of the КМП-3 (KMP-3) machine. It is driven through multidisc friction clutch FC_3, a bevel gear train, and a three-link leverage system. In all other respects operation of the cutting head does not differ from that of the КМП-2 (KMP-2) machine.

Instead of a controller, the MA-191/106к electric motor of the KMP-3 machine uses a КМП-3-110 (KMP-3-110) emergency isolator which has enabled a better rate of cooling to be obtained and allowed the 60-min rating of the motor to be raised to 50 kW and the continuous rating to 22 kW.

The КМП-3 (KMP-3) machine is controlled by means of a three-button station ("Forward", "Reverse" and "Stop") and a ПМВР-1451А (PMVR-1451A) reversing magnetic starter box.

The lubrication chart of the КМП-3 (KMP-3) machine is given in Fig. 75. Table 5 contains a list of the lubrication points and the grades of oil and grease to be used at each point.

The ПМГ-2 (PMG-2) and ПМГ-3 (PMG-3) coalcutters are intended for work in seams 0.4 metre or more thick, and have a body of 310 mm height.

The haulage unit of the ПМГ-2 (PMG-2) machine has two pulsating mechanisms for operating the haulage-speed and flitting-speed multidisc friction clutches, and the haulage rope drum is driven through a worm reduction. Owing to the clutch control, the ПМГ-2 (PMG-2) machine can operate at both variable haulage speed and variable flitting speed, a feature not incorporated in the other coalcutters.

Referring to Fig. 76, the motor shaft extension carries a half-coupling *1* which transmits rotation through an intermediate coupler *2* and a half-coupling *3* to the gear *4*. The latter, in turn, drives the gear *5* and the shaft of the flitting-speed friction clutch *FFC*. Rotation of this shaft is transmitted by a pinion *6*, gear wheel *7*, and pinion *8* to the gear wheel *9* which carries a bevelled-face disc or cam.

The gear wheel *9* is keyed to the shaft of the "feed" or haulage friction clutch *HFC* which, in turn, is attached to a box-section gear *11*.

On a cutting run, the roller *R* of the differential pulsator *DP* will ride round the cam of the gear *9* so that the haulage friction clutch *HFC* is engaged at regular intervals of time.

When the clutch *HFC* is held engaged, the rotary motion of its shaft is transmitted to the box-section gear *11* permanently joined to a worm *12*, and the motion is further transmitted through the wormwheel *13* and internal-tooth gear train *14* and *15* to the haulage rope drum. At the same time, the box-section gear *10* runs idle and the flitting clutch *FFC* is disengaged.

Table 5

Lubrication point (Fig. 75)	Part or point to be lubricated	Lubricant	Periodicity
1	Switching lever rubbing surfaces	AK-15 oil (motor oil 18)	At beginning of each shift
2	Interlock pin rubbing surfaces	Ditto	Weekly
3	Roller and locking sector	Ditto	Weekly
4	Haulage-unit reduction gear	Ditto	At beginning of each shift
5	Motor pinion and follower gear	Ditto	Ditto
6	Electric motor bearings	УTB (1-13) [UTV (1-13)] grease	During every overhaul
7	Cutting head and gummer reducing gear	AK-15 oil (motor oil 18)	At beginning of shift
8	Rubbing surfaces where cutting-head speed switching tie rod enters	Ditto	Weekly
9	Rubbing surfaces of cutting head switching lever	Ditto	Weekly
10	Guide rollers	УCc-2 grease (USs-2 synthetic solidol grease)	Every 15 days
11	Central guide roller	Ditto	Ditto
12	Cutting chain	УH (UN) grease	At end of each shift
13	Nut and bearing of tensioning screw	Ditto	Weekly
14	Gummer reduction gear	AK-15 oil	At beginning of shift
15	Jib head	Ditto	Every 15 days

Note: The cutting chain, and the nut and bearing of the tensioning screw may be oiled with used AK-15 oil instead of УH (UN) grease.

On a flitting run, rotation of the motor shaft is transmitted through the half-coupling *1*, intermediate coupler *2*, shaft and half-coupling *3*, pinion *4*, gear wheel *5*, the shaft of the flitting-speed friction clutch *FFC*, the compressed friction discs of the clutch, box-section gears *10* and *11*, worm reduction *12-13* and the internal-tooth gear train *14*
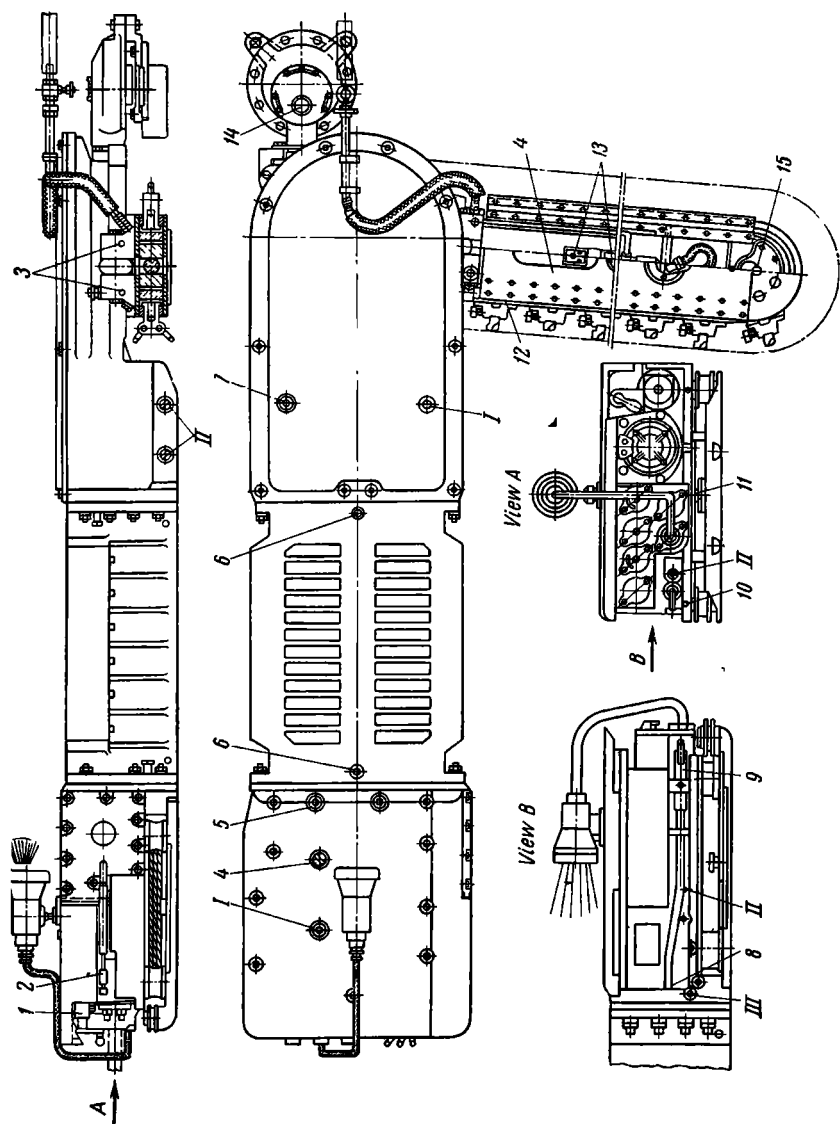
Fig. 75. Lubrication chart of КМП-3 (KMP-3) coalcutter:
I—breather plug; II—drain hole plugs; III—oil-level check plug

Gummer

Drive sprocket

MAD-191/35

7

6

3 2 1

3.3'

18

16.16'

17

5.5'

4.4'

9

8

DP

TR

R

HFC

10

FFC

11

12

13

14

15

20

19

CH
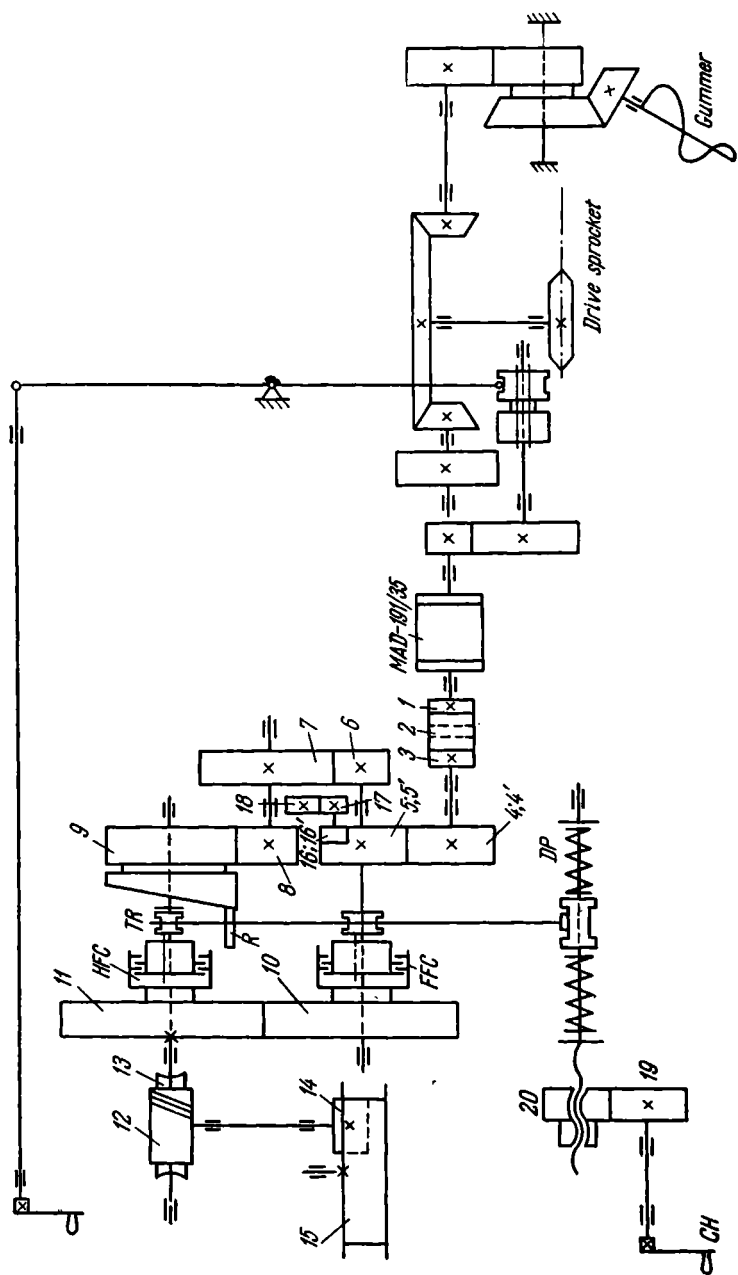
and *15*. At the same time, the haulage-speed friction clutch is disengaged and the gear trains *6-7* and *8-9* run idle.

The oil pump gears *17* and *18* are driven by the gear *16* arranged to engage the gear *5*.

One of the features of the ПМГ-2 (PMG-2) machine is that a switch-over from cutting to flitting or vice versa is effected by reversing the haulage rope drum, and there is no need for reversing the drive motor by the controller, as in other machines.

Another feature of the ПМГ-2 (PMG-2) machine is infinitely variable speed control for both cutting and flitting. A constant flitting speed can be obtained by turning over the thrust ring of the cutting-speed friction clutch shaft.

In intermittent tramming, the roller R rides on the bevelled face of the gear *9*; in flitting at constant speed it bears upon the thrust ring TR.

When the machine is switched over from a cutting to a flitting run, rotation of the handwheel HW is transmitted via the gears *19* and *20* to the shaft of the pulsator DP.

When the pulsator shaft is shifted to the right, the haulage-speed friction clutch is thrown in. If the shaft is shifted to the left, the flitting-speed friction clutch is thrown in. In the neutral position of the control, both clutches are out of engagement and the haulage drive runs idle.

The duration of the interval during which the friction clutches are held in and, hence, the mean speed of cutting or flitting depends on how far the pulsator control shaft has been shifted away from its neutral position.

If gears *4*, *5* and *16*, having 11, 25 and 11 teeth respectively, are replaced by gears *4'*, *5'* and *16'*, with 14, 22 and 14 teeth, the maximum haulage speed is raised from 0.97 to 1.4 m min, and the flitting speed from 7 to 10.2 m/min.

The Cutting Head. The gear box of the cutting head is fitted with a cover having two oil filling holes closed by screw plugs. The jib assembly is put on a bar arm clamped at its rear end between the upper and lower saddles.

The two saddles can be turned through 95 degrees, both ways, by means of the haulage rope. The upper saddle is furnished with two pulleys for this purpose. The bottom saddle has four and the upper saddle has two holes for fixing the cutting jib in the working or the travelling position.

The locking arrangement consists of a spring and lock pin, the latter having a spiral slot, the start and finish of which terminate in longitudinal slots to receive a stop catch.

The haulage unit is jointed to the electric motor by means of seven studs. All the gears of the haulage unit are housed in a cast steel
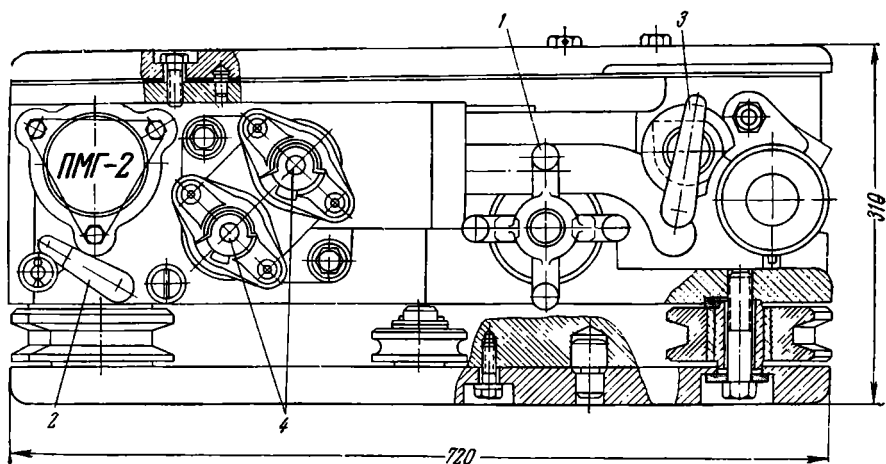
Fig. 77. Controls of ПМГ-2 (PMG-2) coalcutter

Level checking plug

Oil drain plug

Level checking plug

Key:

○ Points inaccessible for lubrication after assembly

◆ Points accessible for lubrication after assembly
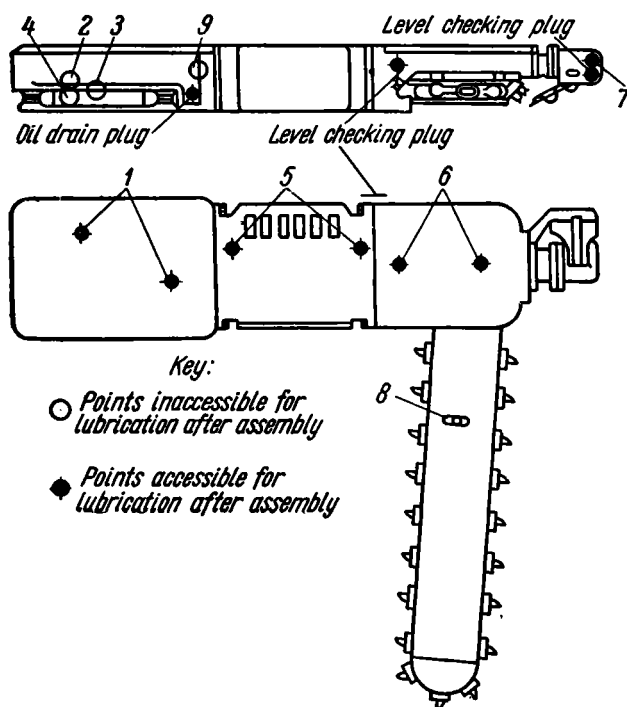
Fig. 78. Lubrication chart of ПМГ-2 (PMG-2) coalcutter

casing closed at the top by a cover. A rubber gasket seals the joint between the cover and the casing. An undercover is attached to the casing by bolts. It carries the rope drum and the rope guide pulleys.

All the control levers of the machine are mounted on the front side of the haulage unit. The control handwheel *1* (Fig. 77) serves to switch in the haulage unit and set the speed of the machine. When turned clockwise, it selects the haulage speed; when turned counterclockwise, it selects the flitting speed. As the angle of turn is increased, the speed rises.

The handle *2* starts or stops the cutting head. The handle *3* switches the controller on or off. The push buttons *4* are for remote control of the gate-end magnetic starter.

The ПМГ-2 (PMG-2) machine is also furnished with a wetting device for dust suppression.

The electrical equipment of the ПМГ-2 (PMG-2) machine includes a МАД-191 35 (MAD-191 35) electric motor with silicone insulation, a КРВ-3006Б (KRV-3006B) controller, a ШВ-10 35 (ShV-10 35) cable coupler, and КУВ-6021А (KUV-6021A) push button stations.

The salient features of the ПМГ-3 (PMG-3) machine are as follows: the haulage unit uses a globoid worm reduction in place of the conventional worm reduction; instead of two springs in the pulsator (one for haulage and the other for tramming), one spring working in two directions is used; the oil piping system and oil seals have been improved; the change gears used make it possible to obtain haulage speeds from zero to 1.38 m min and a flitting speed of 10.2 m min; the machine is driven by a new electric motor, type ЭДК-3-2 (EDK-3-2), with a continuous rating of 14 kW and a 60-minute rating of 42 kW.

A new modification of the above coalcutter is the ПМГ-4 (PMG-4) machine. It is driven by a motor of still higher rating, and its jib and cutting chain are of more advanced design.

A lubrication chart for the ПМГ-2 (PMG-2) and ПМГ-3 (PMG-3) machines is given in Fig. 78. The lubrication points and the grades of oil and grease required for them are listed in Table 6.

The Ural-33 coalcutter. The Kopeisk Engineering Works has developed this new machine for 0.7-metre or thicker seams in order to supersede the КМП-3 (KMP-3) coalcutter.

A distinctive feature of this machine is the use of hydroelectric drive for the haulage unit, with manual (in the Ural-33), or automatic (in the Ural-33A) haulage-speed control. The jib can be mechanically run in and turned by means of the hydraulic drive in the cutting head unit.

Fig. 79 shows the gearing and hydraulic system of the machine fitted with automatic haulage-speed control.

Table 6

Lubrica-tion point (Fig. 78)	Part or point to be lubricated	Lubricant	Periodicity
1	Common oil bath of haulage unit	AK-15 oil (motor oil 18)	Checked at beginning of shift for proper level, oil added if required
2	Lower bearing of vertical shaft	УTB (1-13) [UTV (1-13)] grease	Stuffed when assembled at works
3	Rope drum antifriction bearing	Ditto	Ditto
4	Rope drum free space	УCA (USA) graphite grease	Ditto
5	Motor antifriction bearings	УTB (1-13) [UTV (1-13)] grease	Added every work cycle
6	Cutting head reduction gear	Auto-tractor transmission summer oil	Checked at beginning of shift for proper oil level, oil added if required
7	Gummer reduction gear and gummer supports	Ditto	Ditto
8	Cutting chain tensioning device	Ditto	Every cycle of work
9	Haulage unit coupling	УTB (1-13) [UTV (1-13)] grease	Charged when assembled at works

The prime mover is an electric motor *EM* which drives a hydraulic variable-displacement pump *HP*, capable of delivering up to 150 litres of working fluid per minute to a hydraulic motor *HM*. By adjusting the rate of delivery of the hydraulic pump *HP* with a manual control handle *MCH*, or automatically controlling it by means of an automatic regulator, the speed of the hydraulic motor can be varied between zero and 250 rpm and thereby obtain haulage speeds ranging from zero to 2.4 m/min.

The rotary motion of the hydraulic motor *HM* is transmitted to the haulage rope drum through the haulage reduction gear. A change-over from haulage to flitting speed is effected by means of a haulage control handle *HCH*. The flitting speed can be varied from zero to 6.8 m/min.

When the load on the motor increases, the current flowing through the current transformer *CT* likewise increases, and so does the current
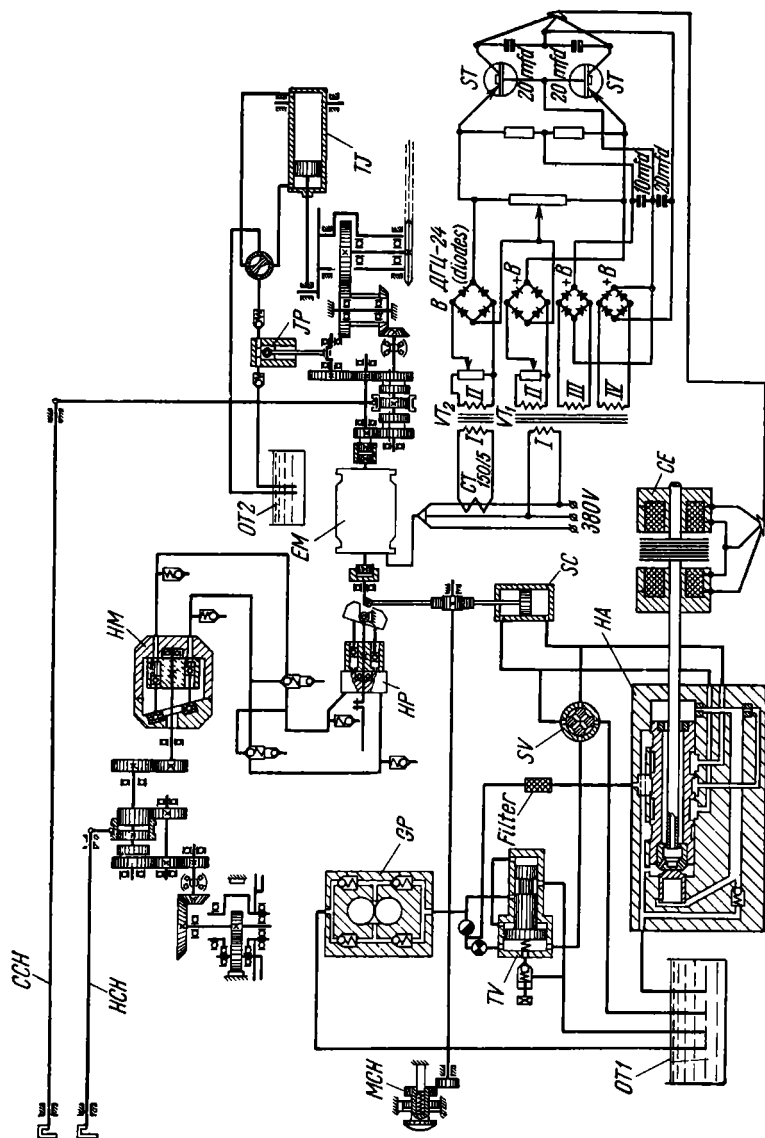
Fig. 79. Gearing and hydraulic circuit diagram of Ural-33A coalcutter

passing through the coils of the control electromagnets *CE*, excited by d.c. supplied by bridge-connected rectifiers *B* and semiconductor triodes *ST*. The coils in the electromagnets *CE* thus control operation of the hydraulic amplifier *HA*.

The control system gear pump *GP* feeds oil via the throttle valve *TV*, its regulator pilot valve, and the hydraulic amplifier *HA* to the automatic control servocylinder *SC*. The piston of the latter moves to reduce the delivery of the hydraulic pump *HP*. As a result, the speed of the hydraulic motor *HM* decreases, and the haulage speed is brought down. The decrease in haulage speed then reduces the load on the electric motor *EM*. The change of current in the electromagnets following the drop in load raises the delivery rate from the hydraulic pump *HP* and, hence, the speed of the hydraulic motor *HM*. In this way the load on the electric motor *EM* is maintained nearly constant.

The hydraulic system in this machine operates at a pressure of 20 atm (gauge).

The valve *SV* effects a switch-over from manual to automatic control. The cutting speed can be changed by shifting the cutting control handle *CCH*.

The cutting jib is run in and turned by a hydraulic drive consisting of pump *JP* and a hydraulic turning jack *TJ*.

The hydraulic diagram shows two oil tanks, *OT1* of the haulage drive system and *OT2* of the jib control.

The haulage drive is arranged in two casings, one housing the variable-displacement hydraulic pump and the hydraulic motor, the other the reduction gear.

As is the case with the КМП-3 (KMP-3) machine, the Ural-33 uses an emergency stop switch in place of a controller.

The Ural-33 has sprayers for dust control and also a lighting fitting for local illumination.

Hydraulic haulage drive offers several advantages over ratchet or pulsating drive. First, it provides infinitely variable speed control for both haulage and flitting. Second, more robust reduction gearing requiring only small space may be used. Third, variations in the load are reduced since the haulage rope drum rotates at a steady speed. Fourth, the haulage drive parts are better protected from overloads. Fifth, the haulage rope drum can be reversed on the run without reversal of the electric drive motor. The hydraulic drive also reduces the noise level of the coalcutter.

The Ural-33 machine is equipped with a type ЭДК4-1 (EDK4-1) electric motor having a 60-minute rating of 88 kW and a continuous rating of 38 kW. Owing to the high rating of its motor, the machine can be fitted with two jibs. It can also operate as a cutter-loader.

Rope drum

To right

To left

Distributor cock positions

Relief valves

Non-return valves

Drain to tank

Jib turn to right

Cock in position
for left turn of jib

Relief valves
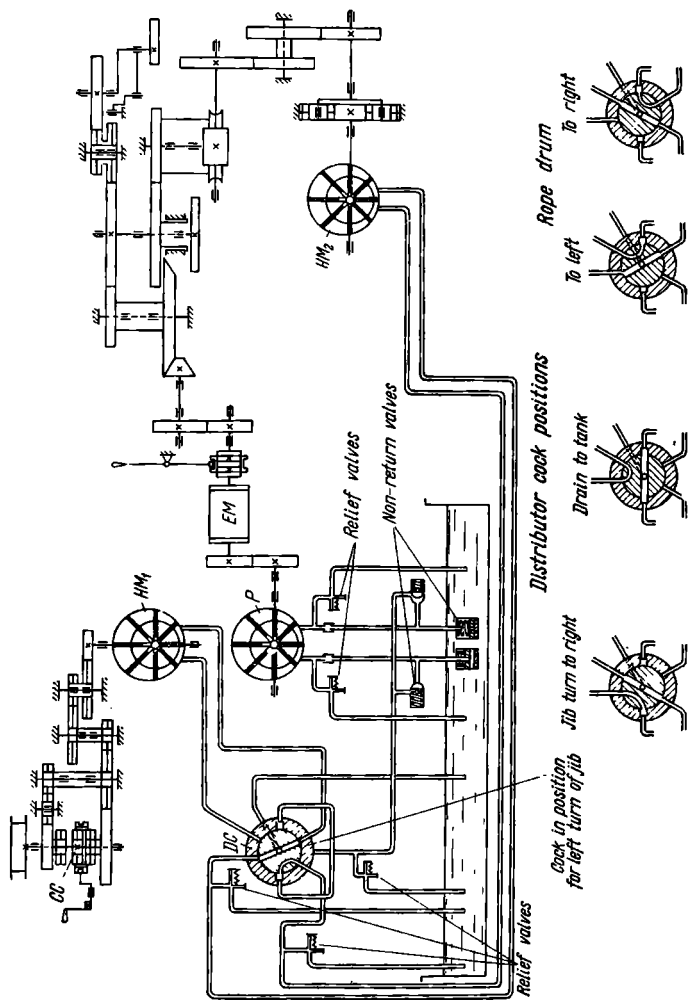
Fig. 80. Gearing and hydrauiic circuit diagram of MBГ-2 (MVG-2) coalcutter

The **MBГ-2 (MVG-2) coalcutter** is intended for work in seams not less than 0.7 metre thick and is equipped with a **ГПЧ-2 (GPCh-2)** hydraulic-drive haulage unit. Fig. 80 shows the gearing and hydraulic system of the machine. The electric drive motor *EM* is fan-cooled and drives a variable-delivery vane-type pump *P*. The pump *P* feeds oil to a hydraulic motor HM_1 which drives the haulage rope drum through a reducing gear. By means of a claw clutch *CC*, the gearing can be made to run at haulage or tramming speed.

In the MBГ-2 (MVG-2) machine the haulage rope drum is top-mounted so that it can hold a rope up to 50 metres long and the rope can be more easily watched as it winds up on the drum.

The haulage unit casing in this machine fulfils two functions; it serves as an oil bath for the gearing and as an oil supply tank for the hydraulic system.

Haulage speed is controlled by varying the eccentricity of the pump. This is done by shifting its body with a screw which ends in a shank fitted with a gear. The gear is turned, as needed, by a rack. The rack is operated by handles acting through gearing.

The distributor cock *DC* directs oil from the pump to the haulage-drive hydraulic motor HM_1 or to the cutting-head hydraulic motor HM_2. An independent facility is incorporated for mechanical turning of the jib.

The **B-32 (V-32) coalcutter** has been designed to supersede the ПМГ-3 (PMG-3) machine and has a body 310 mm high, which fact makes it suitable for seams upwards of 0.45 m thick.

Its haulage unit employs a hydraulic drive consisting of a variable-delivery pump and a radial-piston type hydraulic motor. A mechanical jib-turning mechanism operated by a hydraulic actuator is also provided.

Haulage speed is variable from zero to 3.0 m/min. Flitting speed is 9 m/min. The electric motor ratings are 32 kW, continuous duty, and 42 kW, 60-minute duty, and the main dimensions of the machine are: length 3300 mm, width 720 mm and height 310 mm.

5-17. Cutter-loaders and Shearer-loaders

In default of efficient cutter-loaders for winning the very hard and viscous or sticky varieties of coal and anthracites, or for coping with seams separated by hard stone partings, such coals are in many cases mined by a combination of machine coal cutting and shot firing. Higher face outputs under the above conditions may sometimes be obtained by "shot-fire" loading and use of cutter-loaders.

When loading by shot firing is practised, an armoured or heavy-duty scraper-chain conveyor is set up as close as possible to the face. After a preliminary undercut has been made by machine, the seam

is shot-fired and the main portion of the coal falls directly onto the conveyor. The remaining coal is then shovel-loaded by hand. The machines used to load conveyors include B-7 (V-7), ВНГ-1 (VNG-1), and ВНМГТ (VNMGT).

The B-7 (V-7) cutter-loader is designed to cut and then load coal onto a conveyor at longwall faces in flat seams not less than 0.7 metre thick where roof areas of at least 10 square metres can be left without support.
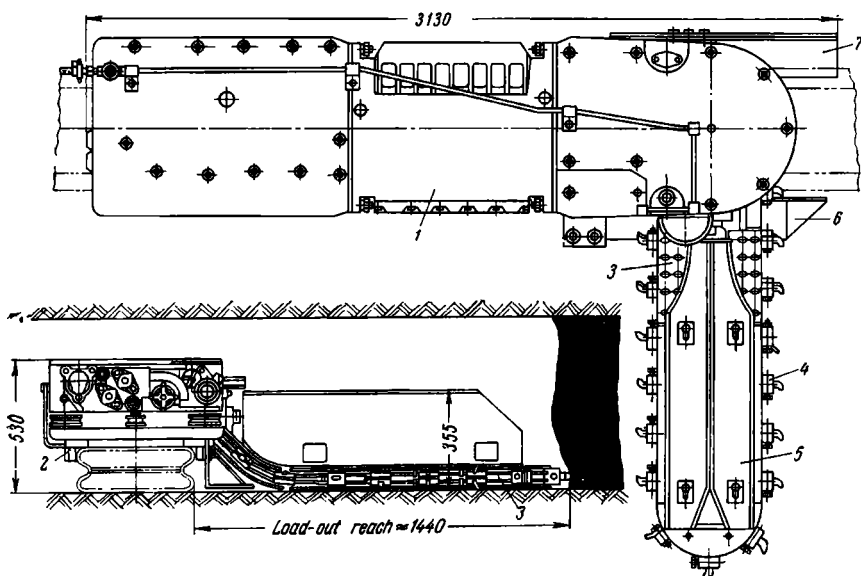
Fig. 81. ВНГ-1 (VNG-1) cutter-loader:
1—cutter-loader; *2*—skids; *3*—cutting head with offset cutting jib; *4*—double-hinged cutting chain; *5*—ramp; *6*—loading ploughshare; *7*—gumming pusher

The B-7 (V-7) machine is hauled along the face on the conveyor frame and has a Z-shaped or offset cutting jib. When the coal is to be loaded onto the conveyor, an apron is fitted on the jib to prevent the coal from dropping back through the jib frame.

At first the B-7 (V-7) machine operates as a conventional coalcutter and makes an undercut along the entire length of the face. After the coal has been broken down by shot firing, the jib is fitted with the loading apron, and the machine hauled backward along the face to load the coal onto the conveyor. For better loading, the cutting chain is fitted with several flight plates. The offset jib efficiently deals with any coal left in the floor.

The ВНГ-1 (VNG-1) cutter-loader (Fig. 81) is intended for work in flat seams from 0.85 to 1.5 metres thick. The coal is loaded onto the

conveyor on the backward run after the machine has undercut the seam and the coal is broken down by shot firing in sections 10 to 20 metres long. An inclined loading apron is fitted on the Z-shaped cutting jib of this machine.

The **BHMГT (VNMGT) cutter-loader** is employed as the two above machines are and is applicable to seams 0.8 metre or more thick. Its haulage unit incorporates a ГПЧ-2 (GPCh-2) hydraulic drive.

The **ГКД-3 (GKD-3) and ГКД-4 (GKD-4) machines** cut and break down soft and medium-hard coals in pitching seams ranging from 0.5 to 0.9 metre thick where the side rock is sufficiently self-supporting. These machines are operated in conjunction with a ГКД-1 (GKD-1) winch mounted in the air gate and are remotely controlled by the machine operator.

The ГКД-4 (GKD-4) machine (Fig. 82) essentially comprises a motor *1*, cutting-end gear box *2*, a flat jib and cutting chain *3*, and a ripping-disc head mounted in cantilever fashion on the flat end of the chain jib. This machine is built around the cutting head of the ПМГ-2 (PMG-2) coalcutter.

These machines work from bottom up. As soon as a web of coal has been cut out, props are set up before a new cycle is begun.

Fig. 82. ГКД-4 (GKD-4) cutting-ripping machine

The ГКД-3 (GKD-3) machine is available only with air motor drive, but the ГКД-4 (GKD-4) machine can have either electric or air motor drive. A further difference is that the body of the ГКД-4 (GKD-4) machine is narrower and weighs less.

Depending on the side rock stability, these machines may work

Troubles in Coalcutters, Their Causes and Remedies

Trouble	Cause	Remedy
Cutting Head		
Cutting chain breaks frequently	Chain is worn Slack chain	Replace the chain Adjust tension so that slack at drive sprocket does not exceed 30 mm
Machine shifts away from coal face	Dull cutter picks	Replace
	Haulage speed too high for cutting hard spots in seam	Reduce haulage speed
Jib cuts into floor or roof	Improperly mounted picks	Re-mount picks according to lacing scheme and check with truing templet
Pinching of jib	Settling of undercut coal	Drive wedges into kerf
	Insufficient stick-out of picks	Align picks by templet
Oil leaks	Worn packings and seals	Replace packings and seals.
Haulage Unit		
Leakage of oil: At joint between haulage unit casing and adapter collar joining motor to haulage unit casing	Loose bolts	Pull all loose bolts tight
At rope drum	Worn felt seal	Replace seal
At screw plug in bottom cover	Missing or loose screw	Tighten or put in new screw
At upper cover joint	Loose bolts, coal dust or grit in joint	Clean joint surfaces and make joint tight
Rope binds between coiled turns on rope drum	Frayed and kinked rope	Replace rope
Rope breaks during cutting operation	Worn rope; rope diameter is too small for load to be taken	Replace worn rope, change over to a larger diameter rope
	Seizure of friction clutch discs	Adjust pulsator, replace bad discs
Improper operation of haulage drive as jib feeds into face	Worn pulsator slide block	Replace slide block
Haulage drive runs hot	Insufficient or excessive lubrication	Lubricate machine as required

with jibs from 0.95 to 1.8 metres long in the case of ГКД-3 (GKD-3) machine and from 1.125 to 1.925 metres long in the case of ГКД-4 (GKD-4) machine. The ripping-disc head may be from 0.5 to 0.9 metre high. This is attained by fitting the respective number of ripping discs on the head.

These machines cannot be used where the side rock is unstable. They also cannot be used in seams where the coal breaks down in large lumps, as the heavy lumps may cause damage to the supports below.

The ГКД-1 (GKD-1) winch is the unit used to haul the cutting-ripping machine along the coal face and is controlled by its operator from a travelling remote control station. This winch can develop a tractive pull of 6 tons at the rope on a cutting run and 1.5 tons on a flitting run. Its drum can take 130 metres of rope from 20 to 22 mm in diameter. The winch can be driven either by a ПРШ-10В (PRSh-10V) air motor of 10hp at an air pressure of 3.5 kg/cm² and an air consumption rate of 10 m³/min, or by a type ВАД-27 (VAD-27) electric motor of 2.7 kW. The weight of the winch when equipped with an electric motor is 1.81 tons. With an air motor it weighs 2 tons.

5-18. Cutting Picks

Current practice is to fit coalcutters with carbide-tipped or hard-surfaced cutting picks of various types.

With time the picks wear and become dull. When the picks are dull, the load on the drive motor will tend to increase, and the haulage speed will have to be reduced. This means that the output of the machine is lowered. Dull picks, moreover, make the cutting jib move somewhat away from the coal face and thus shorten the effective depth of the cut. These are the reasons why a machine should never be operated with dull cutting picks.

A cutting pick comprises a bar body or holder and a head with the cutting edge or tool. The bar or holder end serves as a means for fitting the pick in a chain pick box in which it is locked with a stop bolt.

The front side of the pick cutting edge makes what is termed as the cutting angle with the direction of pick travel. It generally is from 50 to 70 degrees. The angle between the edge back surface and the line of pick travel is called the back angle, usually ranging from 5 to 30 degrees. The angle between the cutting edge front and back surfaces of the pick is called the edge angle.

For cutting soft and medium-hard coals free from hard inclusions, picks with small edge angles of up to 30 degrees are employed (with sharp-pointed wedge-like tips).

When hard coals and anthracites have to be cut, the edge angles are increased up to 45 degrees while the back angles are reduced.

Cutting picks are tipped with cemented tungsten carbide inserts and are also hard-surfaced with special hard alloys. In the U.S.S.R. the grades of cemented tungsten carbide used for this purpose are: BK-6 (VK-6), BK-8 (VK-8), BK-10 (VK-10), BK-11 (VK-11) and BK-15 (VK-15). The T590 hard alloy is employed for hard surfacing.

A pick tipped with a grade BK (VK) tungsten carbide insert is considered to be dull when it wears down 1 mm. Picks hard-surfaced with the T590 hard alloy are considered to be dull after 4 to 5 mm of wear.

5-19. Continuous Coal Cutter-loaders

A continuous coal cutter-loader is a machine that simultaneously cuts, breaks down and loads coal from the face onto a conveyor. It fully mechanizes the fundamental operations of coal winning, substantially raises the productivity of labour, relieves the burden of the miner and brings mining to a more advanced level. The continuous cutter-loader has originated in the U.S.S.R. It may be of interest to note that coal output by continuous cutter-loaders in the U.S.S.R. has reached the highest level in the world.

Continuous coal cutter-loaders are differentiated as to:

(1) seam thickness:

(a) machines for thin seams from 0.4 to 0.8 m thick;

(b) machines for low-height seams from 0.8 to 1.5 m thick;

(c) machines for medium-height seams of 1.5 m and greater thickness;

(2) seam inclination:

(a) machines for "flat" seams with inclinations from zero to 20 degrees;

(b) machines for semisteep seams with inclinations from 20 to 45 degrees;

(c) machines for pitching seams with inclinations from 45 to 90 degrees;

(3) width of cut:

(a) wide-cut machines with heads making cuts wider than one metre;

(b) narrow-cut machines with heads making cuts from 0.4 to one metre wide;

(c) coal ploughs making a cut up to 200 mm (in thickness).

Continuous cutter-loaders are also grouped on the basis of their direction of movement relative to the coal face, the operating principle of the cutting head, the method of feeding the cutting tool into or along the face, the type of drive, etc.

6*

Continuous Coal Cutter-loaders for Flat Low-height Seams

The **Donbass-1 continuous coal cutter-loader** was the first machine of its kind in the world to attain widespread use in coal mining. Successful operation of the Donbass-1 confirmed the belief of Soviet designers that they were on the right track. This opened the way for the development of new types of cutter-loaders suitable for various mining conditions.

The Donbass-1 machine is intended to work in soft and medium-hard coals found in flat seams 0.8 to 1.5 m thick. Provision of a folding profile and a shearing jib has extended its field of application to include seams up to 2.5 to 2.8 metres thick.

A general view of the Donbass-1 is shown in Fig. 83. The main units of the Donbass-1 are the haulage unit *1*; main drive motor *2*; gear box *3* for transmission of power to the cutting tools, ripper bar and gummer; profiling or loop chain jib *4* incorporating a ripper bar and watering sprays; worm type gummer *5*, and loop scraper loader *6*. The machine cuts a web of coal with the profiling chain so as to separate it from the seam, cuts the web into suitable parts with the disc-mounted picks, and breaks the parts down to suitable size with the ripper bar. The scraper loader then gathers the broken coal with its flights and loads it onto the face conveyor. The cuttings are carried out of the kerf by the cutting chain and are loaded onto the conveyor by the auger-type gummer.

The reduction gearing for the cutting chain, ripping bar and gummer in the Donbass-1 is divided into two independent assemblies housed in separate steel casings.

The first casing contains the first-stage reduction gearing and a power take-off for the gummer drive. The second casing houses the second-stage reduction gearing which drives the cutting chain drive sprocket. It also provides a power take-off for the ripping bar drive.

To bring the machine into position for flitting, the second-stage gear case is swung about a vertical axis to a point where it can be locked with a stop pin.

On a cutting run the cutter-loader (Fig. 84) operates in the following way. Rotary motion of the motor pinion *1* is transmitted through internal-tooth gear *2* and pinion *3* to the faced gear *4* which rotates an eccentric sleeve. The latter swings a crankshaft back and forth by means of a connecting rod. This rocking motion is transmitted by a lever and tie bar to the cheeks of a pawl mechanism between which a drive pawl is pivoted on a pin.

On a forward stroke, the pawl engages a tooth on the ratchet wheel *5* and makes it turn. On a return stroke, the pawl simply slides back over the ratchet teeth, leaving the wheel *5* stationary. The ratchet wheel is thus turned intermittently and nonuniformly.
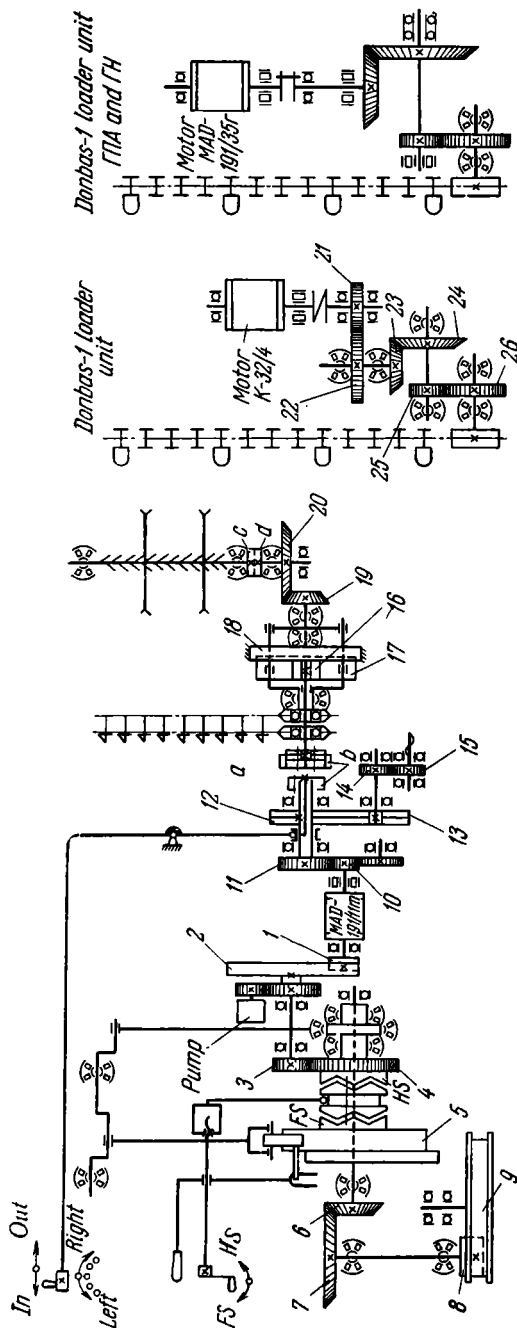
Fig. 84. Gearing diagram of Donbass-1 cutter-loader

Between ratchet wheel *5* and jaw-faced gear *4* there is a clutch which can be shifted along a sleeve seated on the bevel pinion-shaft *6*.

When the claw clutch engages the gear *4*, the bevel pinion-shaft *6* is actuated to transmit motion via a large bevel gear *7* and spur pinion *8* to the internal-tooth ring gear *9* attached inside the rope drum. In this case the rope drum is rotated at a constant speed corresponding to the flitting speed *FS* of the cutter-loader.

If the claw clutch is now shifted to the left, it will engage the jaws of ratchet wheel *5*, and rotary motion will be transmitted through bevel gears *6* and *7* and spur pinion *8* to the drum ring gear *9*.

Drum rotation in this case is intermittent and occurs at the rate proportional to the haulage speed *HS* of the cutter-loader. This speed depends on the position of the feed control plate at the ratchet wheel. When the claw clutch is set in the middle position, no haulage travel will occur.

Gear *10* fixed on the opposite shaft extension of the main drive motor transmits power via gear wheel *11* to a central shaft carrying gear *12*. Actuated by the latter, gears *13* and *14* transmit the power to gear *15* fixed on the sleeve in which the shaft tang of the auger-type gummer is inserted.

The central shaft is hollow and the rod fitted inside it can be moved to shift a half-clutch *b* along the hollow shaft. As the rod moves, the half-clutch goes into engagement with the claws of the opposite half *a* permanently attached to a half-coupling on the end of the central pinion-shaft *16*.

The central pinion-shaft *16* meshes with a set of satellite gears *17* free to ride on stub shafts held in the satellite-carrier yoke. The satellites, in turn, mesh with a permanently fixed ring gear *18*.

When the central shaft is brought into motion, it runs satellites *17* over the teeth of gear *18* and thereby drives the carrier yoke. The latter has a shaft extension on the motor side on which the cutting chain drive sprocket is keyed; on the opposite end the shaft carries bevel pinion *19* meshing with bevel gear *20* which drives the ripper bar and its set of discs and picks through the claw clutches *c* and *d*. Rotation from the motor shaft to the reduction gear of the loader is transmitted by a chain type coupling.

Pinion-shaft *21* meshes with follower gear *22* seated on the same shaft with bevel gear *23* which meshes with a bevel gear *24*. The latter is fixed on the shaft of a spur gear *25* to rotate gear *26* carrying the conveyor chain drive sprocket on its overhung shaft.

The profiling chain jib on the Donbass-1 cutter-loader may be of various height to suit different seam conditions. These heights are:

Seam thickness, metres	Jib height, mm
0.8 to 1.0	710
1.0 to 1.2	830
1.2 to 1.5	1000

Depending on roof stability and coal hardness, use is made of chain jibs with a reach or web cutting depth of 1.6, 1.8 or 2 metres.

In working sticky and hard coal, better breakdown of the upper coal web is achieved by fitting the loop-chain jib with an auxiliary shearing jib mounted on the bracket supporting the ripper bar (Fig. 85).

Several collieries in the Kuznetsk area are successfully employing modified Donbass cutter-loaders in which a folding jib (Fig. 86) is added. Such a jib will fold to reduce its height by 450 mm when the machine is flitted.

The following heights are recommended for the folding chain jib:

Seam thickness, metres	Jib height, mm
1.4 to 1.6	1310
1.55 to 1.7	1430
1.7 to 2.0	1600

The ripper head on the Donbass-1 cutter-loader (Fig. 87) is the unit by which the machine breaks the coal down after the loop chain has cut a web out of the face.

The ripper head consists of a bar *1* carrying cutting discs *3* and a set of spirally arranged cleaver picks *2*. The discs carry pick boxes *5* in which cutting picks *4* are fixed.

The discs *3* are cut away on one side in order to reduce the overall width when the machine is flitted. They are made in two sizes, of 560 and 700 mm diameter. Wherever conditions permit, it is advisable to work with the smaller diameter (560 mm) discs to reduce the load on the motor and the amount of cuttings.

When the coal is friable, one cutting disc is sufficient; in sticky coal seams it is sometimes necessary to fit the bar with three and even four discs.

The loop scraper loaders in the Donbass-1 cutter-loader may be of two types: the ГН1 (GN1) of 656 mm height and the ГП1 (GP1) of 1066 mm height. The ГН1 (GN1) loaders are for working seams up to 1.5 m thick, the ГП1 (GP1) for seams greater than 1.5 m thick.

ГП1 (GP1) loaders are furnished with a coal apron and an attachable foot board on which the propmen can stand in setting props behind the machine in seams greater than 2 metres thick.

Dust control is effected by the water sprayers fitted on the machine.

As soon as a cutting run along the face is completed, the scraper loader unit is detached and swung into line with the longitudinal
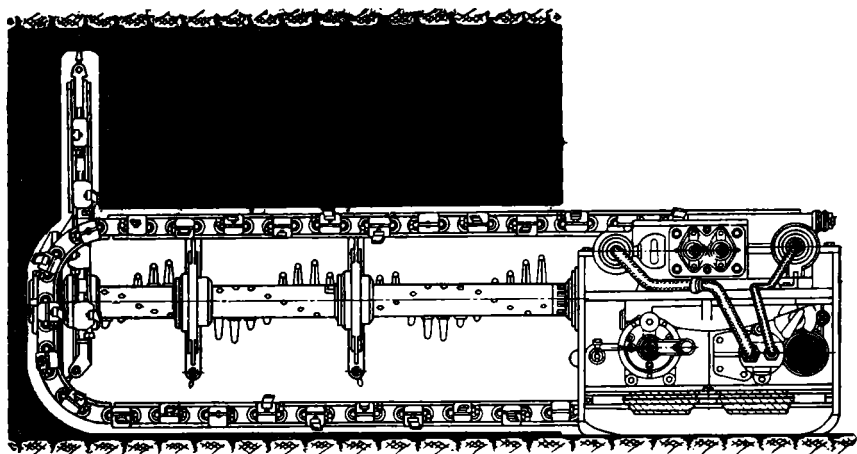
Fig. 85. Loop chain jib fitted with a ripper bar and shearing jib
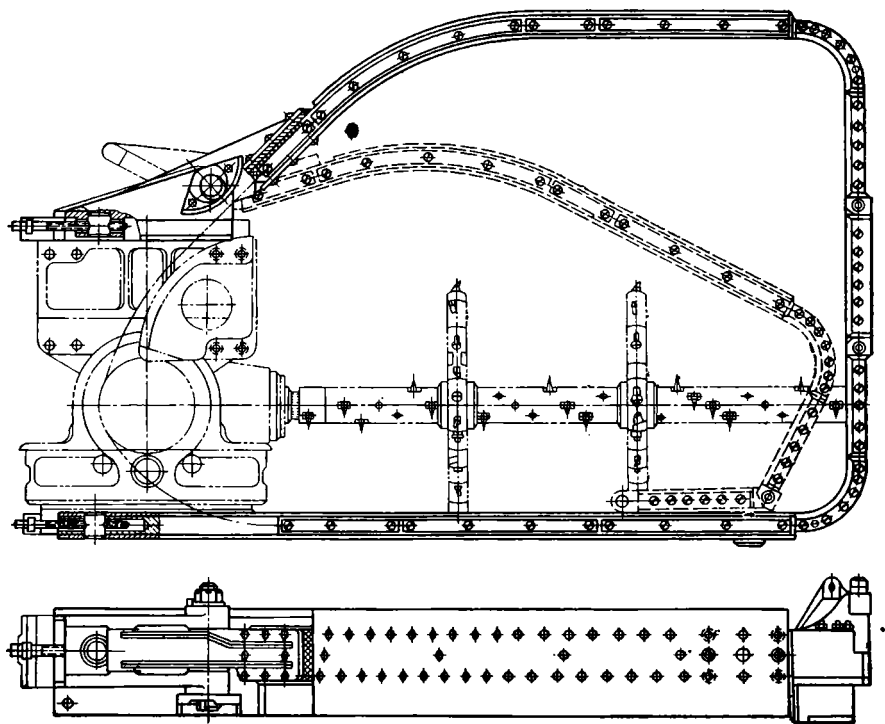
Fig. 86. Folding loop jib

axis of the machine, and so is the cutting head. After being so ar-
ranged, the machine is flitted.

The electrical equipment of the cutter-loader comprises the main
МАД-191/11м (MAD-191/11m) electric motor rated for 65 kW
(60-minute duty), a КРВ-3006 (KRV-3006) controller mounted in
the frame of the main motor, a 32-kW loader drive motor, type
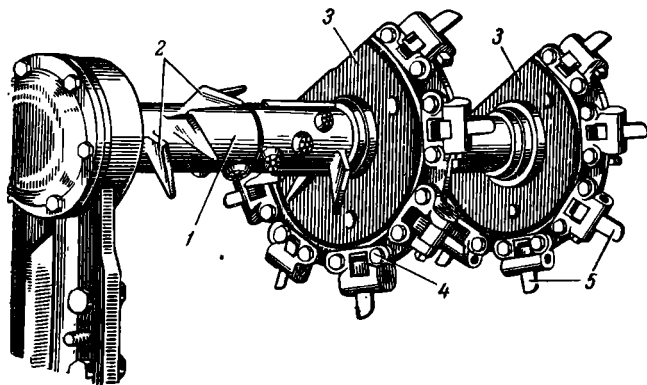
Fig. 87. Ripper head of Donbass-1 cutter-loader

МАД-191/35г (MAD-191/35g), a КРВ-3006 (KRV-3006) loader-
motor controller mounted on the top cover of the haulage-unit cas-
ing, a ШВ-9603А (ShV-9603A) cable connector, two КУВ-6021А
(KUV-6021A) remote-control push button stations mounted on the
haulage-unit casing, a КУВ-6011 (KUV-6011) emergency push
button switch mounted on the loader unit, a 75-VA, 380/36-V
lighting transformer mounted on the controller casing, and a bracket-
mounted flameproof ФВУ-1к (FVU-1k) lighting fitting. The con-
trollers on the machine serve only to reverse the motors, the starting
and stopping being performed by a gate-end magnetic starter box.

The Donbass-1 cutter-loader should be lubricated at the points
shown in the lubrication chart given in Fig. 88 and as prescribed in
Table 7.

The **ЛГД-1 (LGD-1) coal cutter-loader** is a modification of the
Donbass-1 coal cutter-loader using a hydraulic haulage drive con-
sisting of a ГПЧ-1 (GPCh-1) unit and a new ЭДКО-4-1с (EDKO-4-1s)
main electric motor. In all other respects the LGD-1 machine is
similar in construction to the Donbass-1.

The gearing and hydraulic system of the ЛГД-1 (LGD-1) machine
is shown in Fig. 89.

Referring to the diagram, rotation of the electric motor *EM* is
transmitted by gears *1* and *2* to drive a sliding-vane pump *P* whose

Table 7

Ref number in Fig. 88	Total number of points	Assemblies and parts lubricated	Method of lubrication	Interval	Lubricant	Oil bath or lubricant space	
						Weight of lubricant, kg	Oil level or quantity added
1	5	Rope guide pulleys	Grease gun	Grease added every cycle	УТВ (1-13) [UTV (1-13)] grease	0.06	Added till grease shows in clearance gap
2	1	Drum inner space	Put in by hand when assembled at works	Replacement at overhaul	УСА (USA) grease	3.0	—
3	2	Haulage unit oil bath	By splash; circulation by gear oil pump	Added through oil-filling hole every shift	Summer transmission oil (nigrol)	20	To within 170 mm of upper cover
4	1	Haulage mechanism (first stage)	By splash	Ditto	Motor oil 18	5.0	Level with check plug hole
5	2	Main motor bearings	Grease gun	Every cycle of work	УТВ (1-13) [UTV (1-13)] grease	0.6	15 to 20 in squirts

No.	Qty	Name	Method of lubrication	Frequency	Lubricant	Amount	Remarks
6	1	Reduction gear of cutting head and gunner	By splash, oiling gear	When required	Motor oil 18	16	Level with check plug hole
7	1	Gunner reduction gear	Put in by hand	Added every cycle of work	УТВ (1-13) [UTV (1-13)] grease	0.6	—
8	1	Second-stage (planetary) reduction gear	By splash	One litre added each shift	Motor oil 18	5.0	Level with check plug hole
9	1	Cutting head support	Grease gun	Added every cycle of work	УТВ (1-13) [UTV (1-13)] grease	0.3	Added till grease shows in clearance gap
10	1	Loader reduction gear	By splash	When required	Summer transmission oil (nigrol)	7.0	Level with check plug hole
11	1	Bearings No 2314 of loader motor	Placed by hand	Added every month	УТВ (1-13) [UTV (1-13)] grease	0.25	—
12	1	Bearings No. 314 of loader motor	Ditto	Added every three months	Ditto	0.22	—
13	1	Sprayer system pump bearings	Grease cup	Three turns of cap every month	Ditto	0.015	—

suction line is connected to the oil tank (the haulage-unit casing). The valves incorporated in the piping pass the pressure oil through the system always in the same direction, irrespective of the direction of rotation of the electric motor and vane pump P.
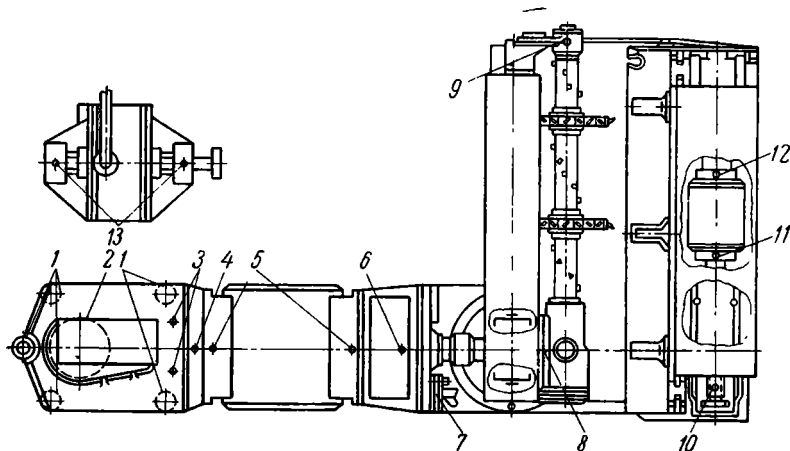
Fig. 88. Lubrication chart of Donbass-1 cutter-loader

If pump P is driven clockwise, as is shown in Fig. 89, the vacuum created in the left part of the pump lifts the valve V_1 to suck oil into the pump from the tank. The pressure the oil attains in the right-hand side of the pump opens the valve V_2 to admit the oil into a distributor D and then into the hydraulic motor HM. The other valves V_3 and V_4 are held closed by the oil pressure.

When the pump is driven counterclockwise, the valves V_3 and V_4 open, while the valves V_1 and V_2 are maintained closed. The oil flow remains unchanged, however, because both valves V_4 and V_2, feeds the oil through the distributor D to the hydraulic motor HM.

Should the oil pressure in the system exceed 32 kg/cm², one of the two relief valves RV_1 or RV_2 (depending on the direction of pump rotation) opens and the oil flows back into the oil tank OT.* One more relief valve RV_3 is included in the oil delivery line to the hydraulic motor and is set for a pressure of 28 kg/cm². A pressure gauge PG is also fitted on this line.

The distributor valve D serves to reverse and stop the hydraulic motor HM.

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* Relief valves RV_1 and RV_2 will only be found in the early series of cutter-loaders; their use was discontinued in the subsequent series.
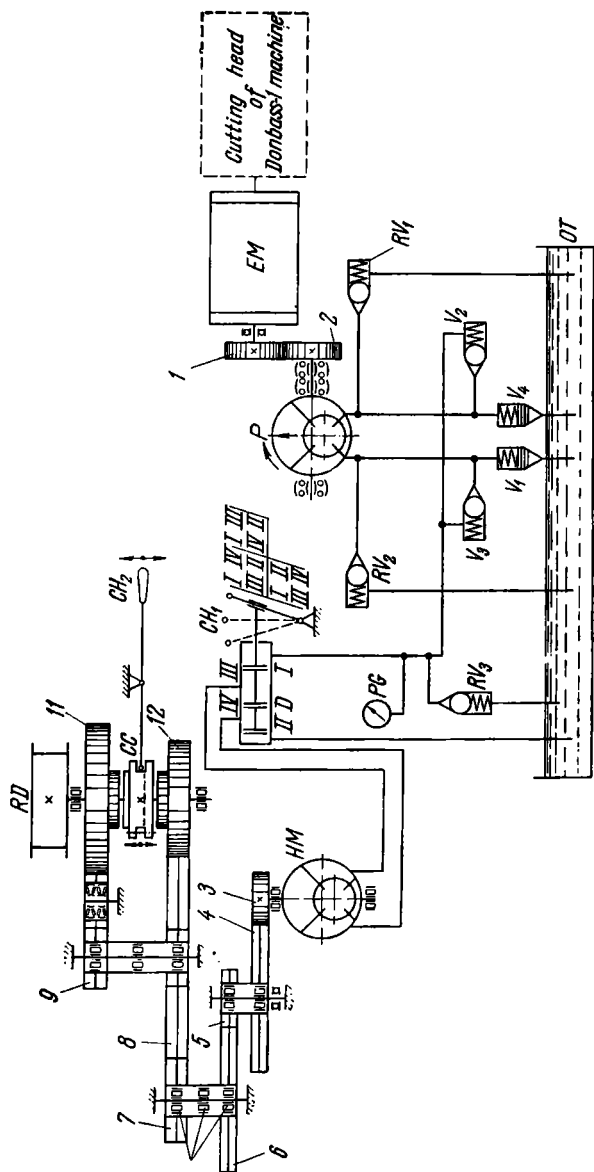
Fig. 89. Gearing and hydraulic circuit diagram of ЛГД-1 (LGD-1) cutter-loader

The valve control handle CH_1 has three fixed positions:

(a) Position *I* in which pressure oil line *I* is connected to line *IV* to feed oil into the left-hand side of the hydraulic motor *HM*, and line *III* is connected to return line *II*. The hydraulic motor *HM* rotates clockwise.

(b) Position *II* in which line *I* is directly connected to line *II*; the oil cannot flow to the motor and is returned to the oil tank. Cutter-loader haulage stops in this position.

(c) Position *III* in which line *I* is connected to line *III* and the oil is fed into the right-hand side of the hydraulic motor *HM*, to leave it through line *IV* connected to line *II* and return to the oil tank. The hydraulic motor rotates counterclockwise.

The shaft of the hydraulic motor *HM* carries a gear *3* which meshes with a gear *4*. The latter transmits rotation through gears *5-6*, *7-8*, and *9-11* to the haulage rope drum *RD* on a cutting run.

On a flitting run, the haulage rope drum is driven by the hydraulic motor through gears *3-4*, *5-6*, and *7-8-12*. A change-over from cutting to flitting or back is effected by means of the control handle CH_2 which shifts the claw clutch *CC* accordingly.

Haulage speed control is attained by changing the eccentricity of the pump casing relative to the pump rotor.

The construction of the pump is shown in Fig. 90. The pump casing *1* houses a rotor *2* seated on a shaft *3*, the extension of which carries a gear *4* which meshes with the gear of the main electric motor. The slots in the pump rotor receive vanes *5* tipped with sliding blades *6*. The blades *6* fit into the groove of the rings *7* supported by end-plate bushes *8*. The end of the screw *9* carries a keyed gear *10* which engages a rack. This rack can be moved by the pump eccentricity-setting handle. The screw *9* turns in the cover plate *11* attached to the haulage unit casing. When the eccentricity-setting handle is moved through some angle, it turns the screw *9* to shift the pump casing upward or downward relative to the rotor. The result is a change in pump delivery, and, consequently, in the rate of oil flow to the hydraulic motor *HM* and in the motor speed.

The pump and the hydraulic motor are similar in construction, but the hydraulic motor is larger in size.

In the ЛГД-1 (LGD-1) cutter-loader the speed of the cutting chain has been increased to 3.08 m/sec; the chain speed of the endless loader has also been increased from 1.54 m/sec in the Donbass-1 cutter-loader to 1.89 m/sec.

A salient feature of the ЛГД-1 (LGD-1) cutter-loader is a top-mounted haulage rope drum and its large rope holding capacity (40 metres).

Donbass-1k Coal Cutter-loader. Since 1960, the Kirov Engineering Works has been making two modifications of the Donbass-1k cutter-
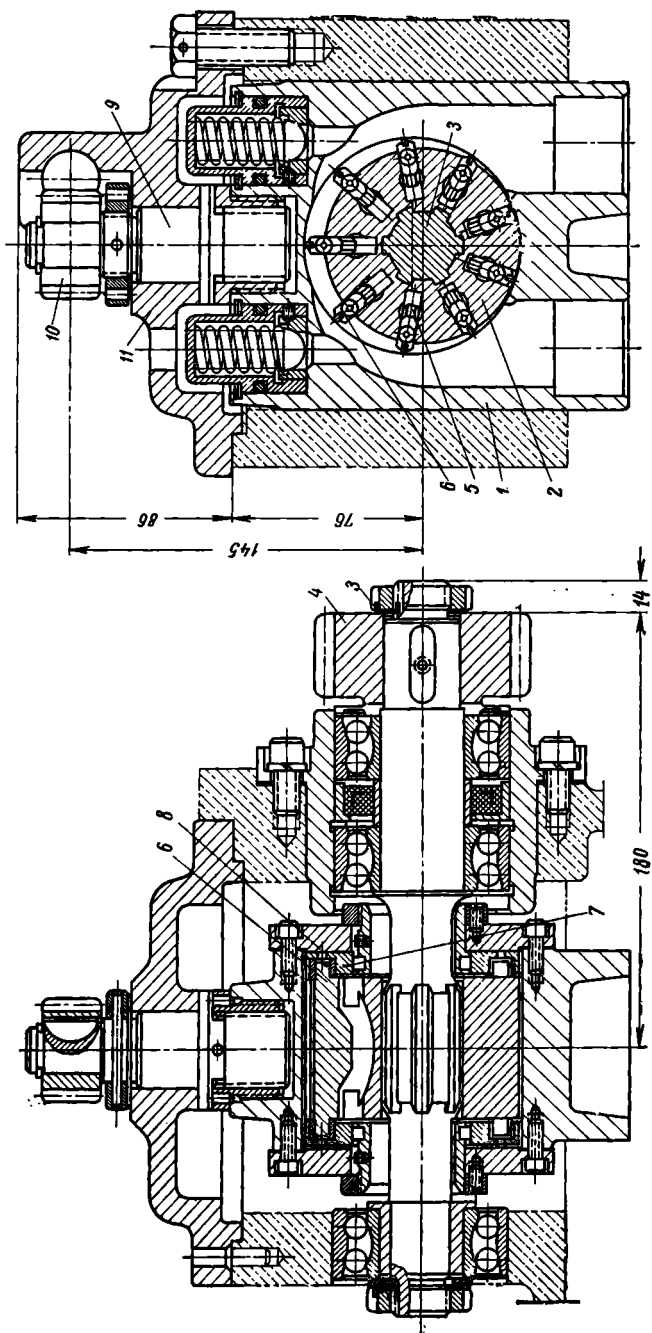
Fig. 90. Features of hydraulic-system pump

Troubles of Cutter-loaders, Their Causes and Remedies

Trouble	Cause	Remedy
Haulage Unit		
Machine fails to operate on haulage speed	Broken working or stop pawl; one of pawl springs broken; broken connecting rod or ratchet wheel teeth, broken ratchet pin	Replace broken part
With handle set in first gear, first speed is not obtained	Worn working pawl (fails to engage ratchet wheel teeth)	Replace working pawl
When speed control handle is set in neutral position, motion continues at haulage or flitting speed	Worn slide blocks on clutch shifting device	Replace slide blocks, set clutch in neutral position
Haulage rope breaks frequently	Rope has lost its strength (broken wires in strands, kinked strands); dull picks; loader jammed between floor and roof; haulage unit has run up against a high spot in face	Replace rope, replace picks, remove all obstacles
Rope jams on drum	Raised spots on rope, rope wound on drum without tension	Replace rope, wind rope only under tension
Cutting-head Gear Box		
Cutting chain does not stop when control handle is set in "Off" position	Worn blocks in clutch control device	Reverse or replace blocks
When control handle is operated, claw clutch will not engage or disengage	Bolt connecting switching tie rod is loose, or screws attaching tie rod to claw clutch have become unscrewed	Tighten loose bolt, drive screws in and lock against loosening
Grease leaks out at labyrinth seal on cutting head side	Worn leather collar	Replace leather collar
Clutch has to be held in place by hand to throw in cutting chain, (clutch does not stay up)	Bad spring	Remove claw clutch and replace spring

Trouble	Cause	Remedy

Cutting and Scraper Chains

Cutting or scraper chain frequently breaks, severe shocks occur during operation	Insufficient chain tension, picks extend too far and catch at gear casing; picks fall out and jam between gear case and chain	Tighten chain, adjust picks for proper height above pick box and fix them tightly in their boxes

Electrical Equipment

Motor hums when switched on, will not start up under load	Blown fuse, burned contacts in magnetic starter, controller, or cable coupler; broken conductor in flexible cable, break in stator winding	Replace fuse, bad contacts; check flexible cable conductors and stator winding with a continuity tester. If break is found to be in motor winding, hand motor over for repair
Motor quickly heats, hums when started, gains speed with difficulty	Rotor rubs against stator because of defective motor bearings	Disconnect motor from drive and try to turn rotor by fan. If rotor turns with difficulty, hand motor over for repair
Motor stalls during operation, runs hot	Low voltage at motor terminals due to excessive drop in circuit	If cutter-loader is in proper condition mechanically, measure voltage across motor terminals. If necessary, take measures to raise voltage to between 360 and 380V (move transformer closer up, change cable for greater size)
	Motor overloaded because of hard inclusions in seam, haulage speed too high, dulled picks and clogged chain	If picks are good, reduce haulage speed. When all stated causes have been corrected, but motor continues to stall, check mechanical drive parts
Machine frame is "live" (shock felt when frame is touched)	One of cable conductors is in contact with frame; earth leakage relay is faulty	Find point of contact with aid of tester and remedy fault. If fault is in stator, hand motor over for repair in surface shop; remedy fault in earth leakage relay

Trouble	Cause	Remedy
Remote control circuit fails to operate (magnetic starter does not close when "Start" button is pushed)	Break in control circuit; bad contact at connection in starter or push button station; burned-out starter operating coil or transformer winding	Eliminate fault
Local lighting fails to light up when magnetic starter closes	Burned lamp; burned lighting transformer in controller chamber of main motor, break in lighting circuit	Check lamp for defects; test lighting circuit for continuity, replace burned transformer

loader; one with a pawl and ratchet haulage and the other with a hydraulic ГПЧ-2 (GPCh-2) haulage drive [the ЛГД-2 (LGD-2) cutter-loader].

As compared with the Donbass-1 machine, the following essential changes have been made in the Donbass-1k cutter-loader using pawl and ratchet haulage: the number of haulage speeds has been increased from four to five (0.33; 0.66; 0.99; 1.32 and 1.65 m/min); a higher flitting speed is provided; the haulage rope drum has a larger holding capacity, and a type ЭДК4-1 (EDK4-1) electric motor is used as the main drive.

The Donbass-2k coal cutter-loader works in flat seams 0.85 to 1.6 metres thick with coals of various hardness and stickiness and where the roof is of medium stability.

This machine is more powerful than the Donbass-1 and can therefore work in seams of medium hardness at raised haulage speeds and show higher output per shift. The Donbass-2k operates similarly to the Donbass-1, the differences being as follows:

(1) The haulage unit in the Donbass-2k is of the friction-pulsating type, its operating principle being the same as in the КМП-2 (KMP-2) and ПМГ-3 (PMG-3) coalcutters. The haulage speed is variable, ranging from zero to 1.5 m/min.

(2) The drive sprocket of the cutting chain is set high and placed closer to the loader conveyor. This eliminates the need for a gummer.

(3) The Donbass-2k is equipped with an ЭДК-120 (EDK-120) electric motor rated for 120 kW, 60-minute duty, and 50 kW, continuous duty. To match this considerably higher power capacity, all the electrical equipment on the machine operates on 660V and the power is supplied to the machine through a screened flexible ГРШСНЭ (GRShSNE) cable.

(4) The ПМВИ-1635 (PMVI-1635) reversing magnetic starter gate-end box is remotely controlled with three push buttons ("Forward", "Reverse" and "Stop"), and the controllers have been replaced by two emergency switches mounted in the haulage-unit casing, one for the main motor, the other for the loader motor.

(5) Being of substantially higher power capacity, the Donbass-2k can work not only with a loop or folding cutting jib, like the Donbass-1 machine, but also with either a rigid or a folding double jib. In the double jib the two arms carrying the cutting discs are set one above the other.

The KP-1 (KR-1) coal cutter-loader has an adjustable cutting head and is intended for extracting medium sticky coal of greater than medium hardness in flat seams from 0.9 to 1.5 metres thick. The KP-1 (KR-1) cutter-loader (Fig. 91) comprises a haulage unit *1*, the same as that employed in the Donbass-2k, a type ЭДК-120 (EDK-120) electric motor *2*, the reduction gear *3* of the cutting-head and gummer, loop chain jib *4*, loop scraper loader *5*, disc-type ripper bar *8*, gummer *6*, horizontal hydraulic actuator *7*, and water sprayers.

The chain jib is only 420 mm high. The hydraulic actuator raises or lowers the disc-type ripper bar to get down any seam of given thickness. The cutting head can be adjusted for height, so that the whole coal seam will be got down without leaving a roof coal web; the machine can easily sweep its ripper bar from top to bottom for flitting.

Coal Cutter-loaders for Thin Flat Seams

The УКМГ-3 м (UKMG-3 m) coal cutter-loader is designed for working very thin flat seams 0.4 to 0.6 m thick lying between medium stable strata and consisting of medium-hard or sticky coals free from hard inclusions.

This cutter-loader is built around the ПМГ-2 (PMG-2) coalcutter to which a remodelled cutting unit has been fitted.

Two jibs are incorporated in the machine (Fig. 92): a lower offset jib *1* with a double-hinge cutting chain, and a straight jib *2* with a single-hinge cutting chain, driven by the lower jib chain through the double sprocket of the ripping disc *3*.

The cutting head can be set for a working height of 0.36 to 0.4 m by replacing the double sprocket and the jib bracket, or for a working height of 0.3 m, or of 0.45 to 0.5 m after a simple refit.

Each cutting chain is kept taut by its own tensioning device.

The direction in which the chains run is opposite to that practised in other machines. The front picks cut into the face, while the rear strands, instead of running idle as in coalcutters, catch the coal and load it onto the face scraper-chain conveyor.
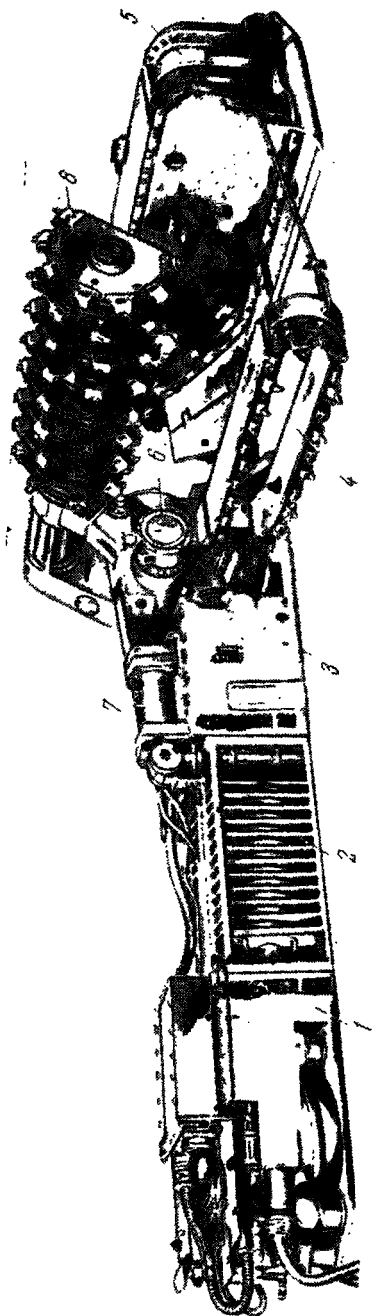
Fig. 91. KP-1 (KR-1) coal cutter-loade r
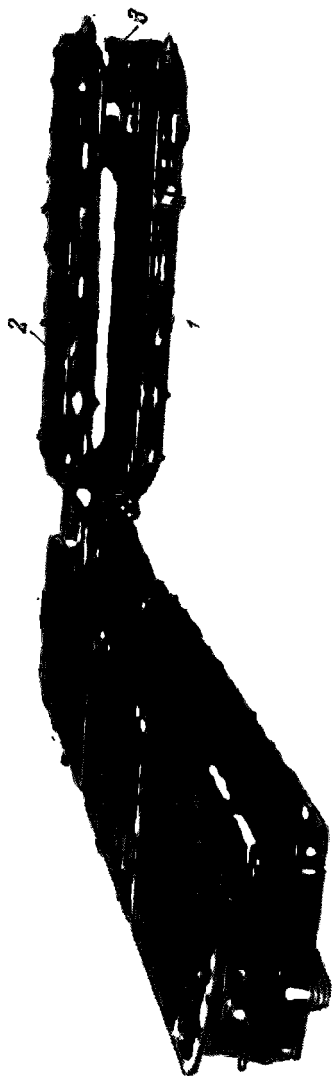
The machine has an inclined loading apron (omitted in Fig. 92) set up behind the jibs and cutting unit within 0.8 m of the front part of the jib assembly. This apron has hinged end plates which give access to the chains for pick replacement.

A screw arrangement incorporated in the off-set jib saddle is used to change the inclination of the jib. This makes it possible to undercut the seam without leaving any coal on the floor.

Fig. 93. КЦТ (KTsT) cutter-loader

Dust suppression is effected by the water sprayer system. It consists of a pipe laid along the edge of the apron and provided with two or three hose outlets carrying water spraying nozzles at their ends.

This cutter-loader is operated in conjunction with CKT$_2$-6 м (SKT$_2$-6m) or KC-10 (KS-10) scraper conveyors. For this, the side or spill plates welded to the conveyor trough must be removed on the face side, and there must be no props set up between the conveyor trough and the cutter-loader.

УKT-2м (UKT-2m) and КЦТ (KTsT) coal cutter-loaders differ only in haulage. The УKT-2м (UKT-2m) coal cutter-loader has a conventional rope haulage drive, while the КЦТ (KTsT) machine operates with haulage along a sized chain strung from end to end of

the face. These cutter-loaders have been designed to extract hard and sticky coals and anthracites lying in thin flat seams from 0.55 to 0.75 m thick between strata of not less than medium stability.

A distinctive feature of these cutter-loaders is that they work on the shuttle principle, i.e., without idle back run. They turn at the end of each cut in stable-holes at the face ends.

The УКТ-2м (UKT-2m) machine is based on the haulage unit and electric motor of the КМП-2 (KMP-2) coalcutter machine; and the КЦТ (KTsT) machine on the haulage unit of the ПМГ-3 (PMG-3) coalcutter and a type ЭДК4-1 (EDK4-1) electric motor.
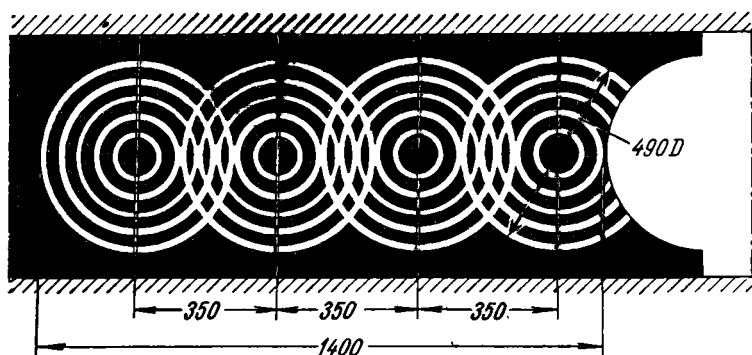
Fig. 94. Cutting pattern of the rotating pick arms of КЦТ (KTsT) cutter-loader

The essential parts of the КЦТ (KTsT) coal cutter-loader (Fig. 93) are the cutting head assembly and gear box, electric motor, haulage unit, raising mechanism, guide skids, left-hand and right-hand side panels, watering sprayers, and a water hose.

In this machine the cutting head has four rotary cutting arms *1* and a loop cutting-loading chain *2* placed back of the arms to fit the contour of the face.

Each cutting arm carries a starting bit and from four to five picks.

As the machine operates, the picks cut deep concentric kerfs in the face (Fig. 94). As the picks dig deeper, the coal between the kerfs breaks down, while the cylindrical core of coal in the centre is knocked off by the starting bit.

From reference to Fig. 94, it can be seen that the circular kerfs left by adjacent arms partially overlap.

For better loading performance, the cutting-loading chain, in addition to its picks, is fitted with several small flights. Part of the coal is loaded by the cutting arms themselves, as they rotate away from the face and towards the conveyor.

Depending on the thickness of the seam to be worked, chain jibs 500, 550 or 625 mm high may be used.

Where a danger of local roof failure exists, a travelling remote-control station can be used.

Haulage of the КЦТ (KTsT) cutter-loader along the chain is performed by the sprocket incorporated in the haulage unit in place of a rope drum. The chain itself is attached to anchor props at both ends of the face. As the cutter-loader works its way down the face, the chain can be moved up to the new face.
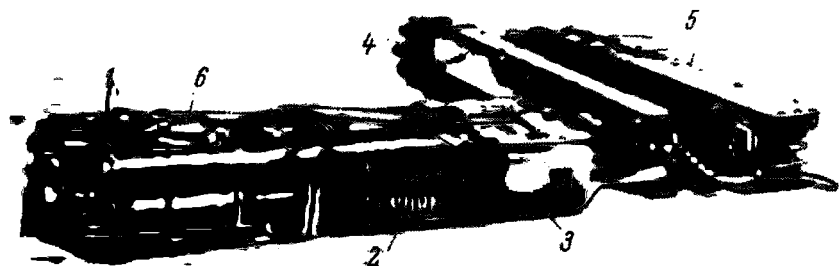
Fig. 95. Gornyak-1 cutter-loader

Positive engagement of the drive sprocket with the chain is ensured by tensioning the chain with a screw jack which can be actuated by an electric drill. Helical springs protect the haulage chain against overloads.

The chain haulage has made it possible to withdraw the machine from the face for pick replacement by simple reversal of the drive sprocket, whereas in the earlier УКТ-2м (UKT-2m) machine this involved difficult manoeuvring. Another advantage is the saving in time for move-up of anchoring props.

The Gornyak-1 coal cutter-loader (Fig. 95) is intended to work flat coal and anthracite seams from 0.6 to 0.8 m thick where the roof can be left unsupported over areas of at least eight square metres. The Gornyak-1 is not suitable for operations in hard, non-cleaving and very tough coals, and also where sticky coal is encountered.

This machine consists of a haulage unit *1*, main electric motor *2*, gear box *3*, and the cutting-end and gummer drive and a turning head. This head supports a loop cutting-chain jib *4*. The loop scraper loader *5*, driven separately by its own electric motor, gathers and loads

the coal onto the face conveyor. The machine is also fitted with watering sprayers for dust control.

A window *6* in the top cover of the haulage unit casing gives access to the pawl and ratchet mechanism of the haulage unit for replacement.

The main ЭДК4-1 (EDK4-1) electric motor of the Gornyak-1 has the two compartments in the frame for the controllers; one houses the main motor controller, and the other the loader motor controller.

The Gornyak-1 is an improved version of the Donbass-1. In fact, the haulage unit is the same in both.

The cutting head of the Gornyak-1 may be either loop or a folding chain jib of the same type as that employed in the Donbass-1, the only difference being that the Gornyak-1 has no ripping bar.

As the machine advances, the web of coal emerging from the loop jib breaks down under its own weight. Depending on the height of the chain jib, the coal web can be either 240 or 340 mm thick.

The loader unit is mounted 0.8 m back of the loop jib. This makes the coal web overhang just far enough to break off at the root.

When the coal falls, it is broken up into more transportable lumps by the loader scraper flights and is loaded on the face conveyor. The flights are furnished with picks to break up the coal more efficiently.

Where tough coals have to be worked, breakdown is aided by mounting the loader as far as 1.7 metres back of the loop jib. This is only permissible in stable beds, because this increases the unsupported area of the roof.

The **Shakhtyor-2 coal cutter-loader** is designed to work 0.5 to 0.8 m seams of soft and medium-hard coal lying in weak beds.

It is based on the КМП-2 (KMP-2) coalcutter, the haulage unit and motor of which have been incorporated without any change.

The reduction gearing which is the same as for the КМП-2 (KMP-2), has two additional helical gears. Through the above reduction gearing and a claw clutch, motion is transmitted to the shaft carrying two sprockets which drive both the cutting and loading chains.

The cutting and loading chains run in the guides of the profiling jib. As a web of coal is cut round the profile, it drops on a metal apron, breaks up, and is loaded by the loading chain scrapers onto the CKT_2-6м (SKT_2-6m) face conveyor.

For better loosening of the coal lumps, the chain flights are supplied with spikes. The mouldboard attached to the apron is set at some angle and aids in coal loading.

A shortcoming of the Shakhtyor is the absence of a ripping head. Since the distance between the cutting and loading chains is rather small, the coal web will at times not break off under its own weight, instead, running against the mouldboard, it partially spills over. This coal has to be loaded onto the conveyor by hand.
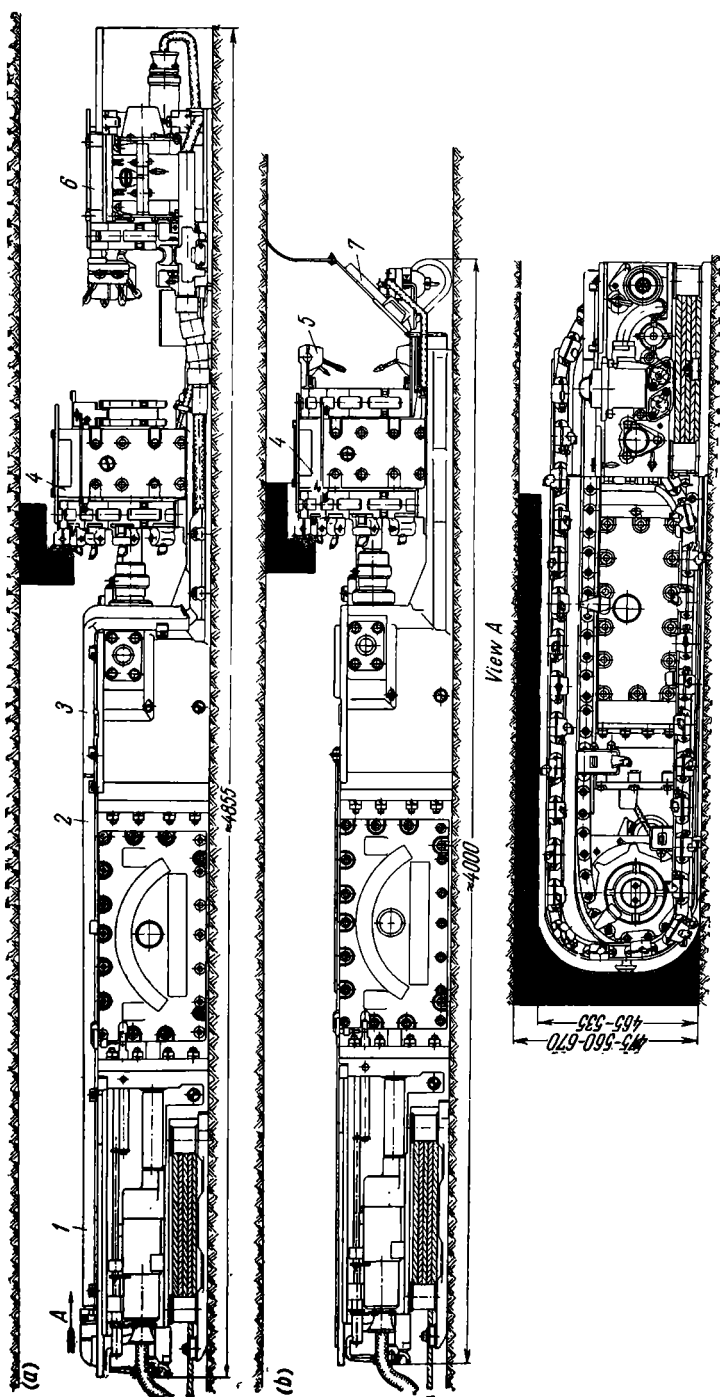
Fig. 96. Kirovets cutter-loader:

1—feed head; *2*—electric motor; *3*—cutting-head reduction gear; *4*—cutting head;
5—loading scraper chain; *6*—loop loader; *7*—loading ramp

The driving sprocket of the cutting chain is placed close to the conveyor. Therefore, there is no need for a gummer; the cuttings are thrown onto the conveyor by the chain itself.

The Kirovets coal cutter-loader is designed to work 0.55 to 0.9 m flat seams of medium-hard coals which are not tough, cleavable, and lie between beds of medium stability.

The Kirovets has superseded the Gornyak-1 and the Shakhtyor. It is manufactured in two modifications and can operate as the Gornyak-1, i.e., with a loop loader (Fig. 96a), or as the Shakhtyor-2, i.e., with a loading mouldboard and flight-loading chain in the rear guides of the jib (Fig. 96b).

In operation with a loading unit, the flight-loading chain 5 and the loading mouldboard 7 are removed and replaced by a loop loader 6 and a guard to protect the loader cable. As an alternative, both the flight-loading chain and the loader can be used simultaneously, a mode of operation that permits the coal cutter-loader to work harder and tougher classes of coal.

The haulage unit of the Kirovets is the same as that of the ПМГ-3 (PMG-3) coalcutter, but the haulage rope drum has been made higher and the casing larger to take up to 50 metres of rope.

The cutting and drive heads in the Kirovets are the same as in the Shakhtyor-2.

The jib of this cutter-loader has two chain guide ways; the cutting chain running in the front guide ways is the same as in the Donbass-1, the rear guide carries a flight-loading chain assembled of Donbass-1 loader chain parts and flights fitted with the cutting picks and cleavers used in the УКТ-2м (UKT-2m) cutter-loader.

The loading mouldboard is attached to the lower cheek of the jib. The mouldboard can fold into the loop jib, thereby reducing the width of the machine for flitting.

The Kirovets can use a loop jib with a cutting reach of 1, 1.65 or 1.8 metres and a height across the picks of 475, 560 and 670 mm, or a folding jib having a cutting reach of 1 to 1.65 metres and a height across the picks of 600, 680 or 770 mm.

Dust control equipment is the same as in the Donbass-1.

The lubrication chart of the Kirovets cutter-loader is given in Fig. 97; Table 8 contains the lubricating instructions.

The K-8 coal cutter-loader is designed to work thin flat seams of hard and tough coals from 0.6 to 0.9 m thick.

It has been built around the haulage unit of the ПМГ-2 (PMG-2) coalcutter. A similar type of cutter-loader, the К-8Д (K-8D), has been constructed on the basis of the КМП-2 (KMP-2) coalcutter.

The cutting head of the K-8 cutter-loader is a rotary horizontal bar fitted with a set of cutting discs. A hydraulic actuator serves to raise the bar to any height required for working the seam.

Table 8

Lubrication point		Part or assembly to be lubricated	Interval	Lubricant
Ref. number in Fig. 97	Number of points			
1	1	Haulage unit worm gear reduction	Oil level to be checked at beginning of every shift (100 mm from top cover); additions as needed	Motor oil 18
2	1	Haulage unit friction clutch	Ditto	Ditto
3	1	Lower antifriction bearing or rope drum	Greased when assembled at works	УTB (1-13) [UTV (1-13)]; grease
4	1	Free space in rope drum	Ditto	УCA (USA) grease
5	1	Upper antifriction bearings of rope drum	Every cycle of work	УTB (1-13) [UTV (1-13)] grease
6	1	Haulage unit coupling chamber	Stuffed when assembled at works	УTB (1-13) [UTV (1-13)] grease
7	2	Cutting head reduction gear	Additions made as needed	Industrial oil 30
8	2	Electric motor bearings	After every cycle of work	УTB (1-13) [UTV (1-13)] grease
9	1	Drive-head reduction gear	Addition of oil at beginning of each shift	Industrial oil 30
10	1	Loader reduction gear	Oil level to be checked at beginning of every shift; additions made as needed	Ditto
11	1	Antifriction bearings in end head of loader	Ditto	Ditto
12	4	Tensioning devices of cutting and loading chains	Every cycle of work	Ditto
13	2	Loader motor bearings	Ditto	УTB (1-13) [UTV (1-13)] grease
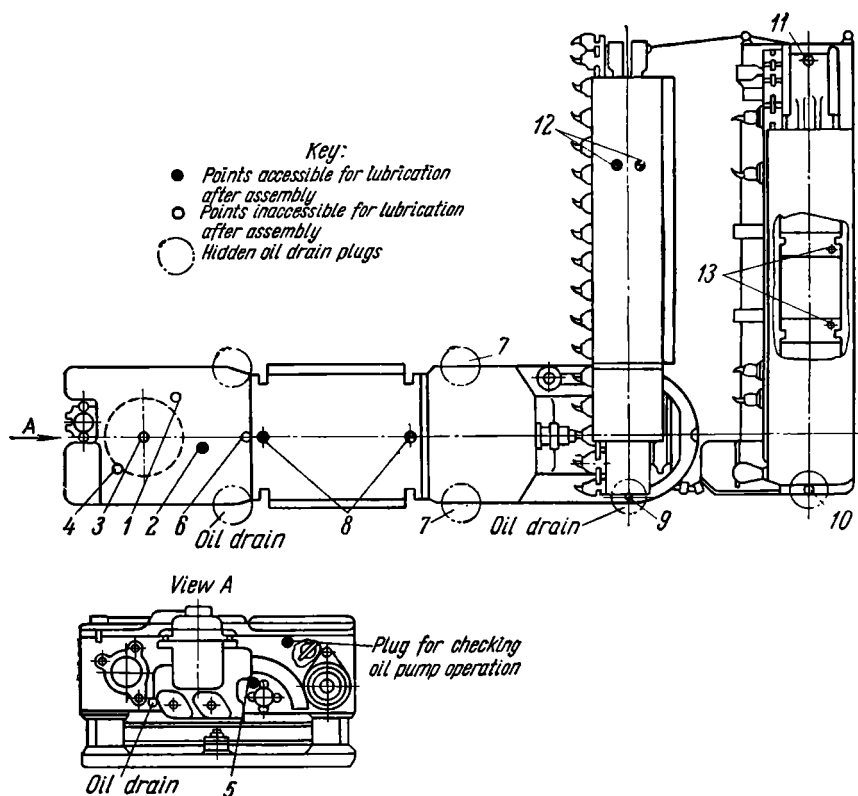
Fig. 97. Lubrication chart of Kirovets cutter-loader

The function of the cutting head is to break down the upper part of the seam, the lower part of the seam being broken down by the loop loader scraper chain driven by the main electric motor.

Continuous Coal Cutter-loaders for Thin and Low-height Inclined Seams

Thin and low-height inclined seams are worked by the same cutter-loaders as are flat seams of comparable thickness. However, in inclined seams the loop chain loader unit or the loading mouldboard are replaced by a chute which diverts the coal onto the conveyor. The chute is attached to the cutter-loader and can be of two types: the face chute which is laid between the face and the first row of roof supports, and the central chute which is laid between the first and second row of props.

When coal cutter-loaders operate in inclined seams, the U.S.S.R. safety regulations require the installation of a winch for the safety rope. The features of such winches are discussed in the section on cutter-loaders used in pitching seams.

A machine especially developed for work in medium-height inclined seams is the **K-8H (K-8N) cutter-loader.**

The **K-8H** (K-8N) (Fig. 98) can work hard and tough coals in semisteep seams of 25-to 50-degree inclination and from 0.7 to 1.15 m thick.
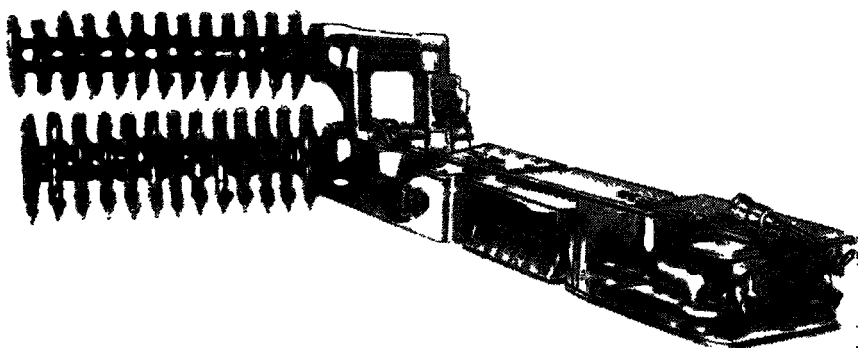
Fig. 98. K-8H (K-8N) cutter-loader

It is built around the haulage unit and electric motor of the КМП-2 (KMP-2) coalcutter. The cutting head is made up of two rotary bars carrying cutting discs. The upper rotary bar assembly can be raised or lowered during operation to suit the thickness of the seam being mined, which fact obviates the need for subsequent breakdown of the upper web. The upper rotary bar assembly is raised and covered by a manually operated worm reduction.

Coal Cutter-loaders for Pitching Seams

According to the mode of operation, the coal cutter-loaders working thin and low-thickness pitching seams can be divided into two groups: machines cutting downward and machines cutting upward.

Downward-working machines permit roof supports to be set up directly behind the machine as it advances downward. However, a significant disadvantage of such machines is that the cutting head

cannot operate at full capacity, because it is fed into the coal face only by the dead weight of the machine.

Upward-working machines require periodic stoppage to set up roof supports. In certain cases, where the coal falls off in large lumps as the cutter-loader advances upward, the roof supports put in just
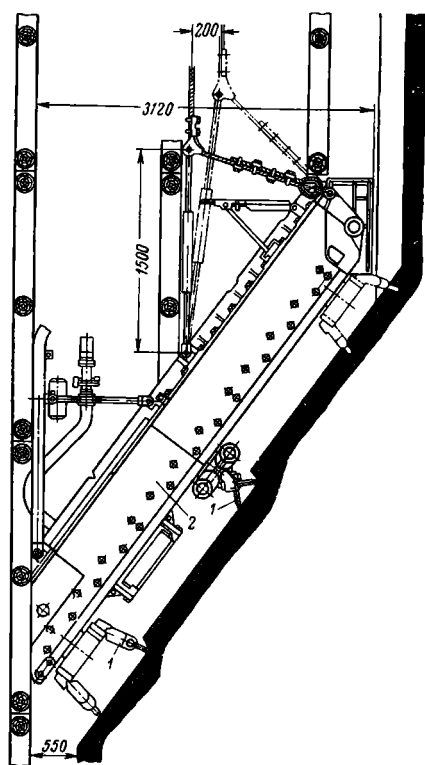
Fig. 99. ККП-2 (KKP-2) coal mining machine
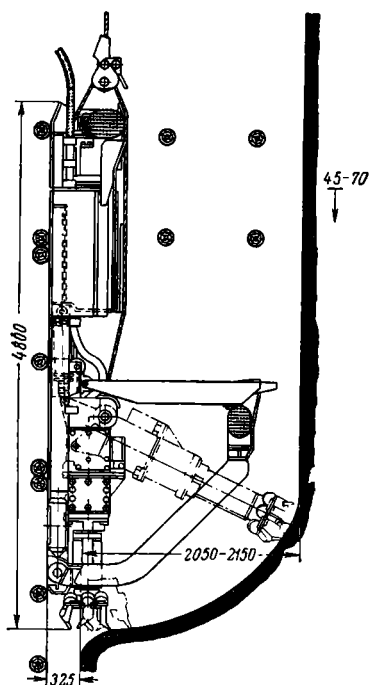
Fig. 100. K-19 coal mining machine

below the machine may be knocked out by the falling coal. Nevertheless, since the cutting head is forced-fed into the face, it can, therefore, operate more efficiently.

Thin and low-height pitching seams are still mined with continuous cutter-loaders on a limited scale, being mainly worked with air picks. Further efforts must be therefore made to develop machines which will fully mechanize mining operations in various classes of thin and low-height pitching seams.

The **KKП-2 (KKP-2) coal mining machine** (Fig. 99) is designed for operation in pitching seams with inclinations of 50 to 70 degrees and from 0.65 to 1.3 m thick.

This machine works from the seam top downward on an inclined bench from 2.1 to 2.2 m wide.

The cutting unit in this machine consists of two rotary pick arms *1* which rip the coal off the face. The two arms rotate on the vertical spindle of the ripper assembly which reciprocates together with its drive motor and gear box in the inclined frame *2* of the machine. The machine also has a seat and safety shield for the operator.

On upward traverse, the ripper assembly arms cut a web of coal up to 200 mm thick from the face. On return (downward traverse) the arms are idle. During the cutting run, the frame of the machine is held immovable by two anchor props on one side and one prop on the other.

The machine is fitted with a skid for control of its movement.

During the return traverse, the anchor props are slightly shifted and the machine is lowered on the rope as much as is necessary for the next cut to be made. Then, the anchor props are again set to hold the machine in position for the new working cycle. The machine is raised and lowered by a winch mounted in the upper level.

The KKП-2 (KKP-2) machine is equipped with a ПШД-30 (PShD-30) air motor of 32 hp. The winch is driven by a ПРШ-16k (PRSh-16k) air motor of 16 hp.

Remote control of the machine and winch is accomplished by means of a distributing valve to which air supply lines are run.

The KKП-2 (KKP-2) can only work under stable roof and floor conditions in soft and semihard coals. The area left unsupported in the working zone of the machine is not less than 8.75 m².

The **K-19 mining machine** (Fig. 100) is intended for work in pitching seams from 0.5 to 0.8 m thick where the roof and floor are of at least medium stability. It can mine coals of any hardness, but not when large pyrite inclusions are encountered in the seam.

Like the KKП-2 (KKP-2), the cutting head of the K-19 is fed into the coal face by the dead weight of the machine. The width of the face bench for this machine is 2 to 2.1 m.

In the K-19 machine the cutting unit is of the rotary crown type, with single or double pick sets. The head travels in a guide framework and is swung across the face by a hydraulic actuating cylinder. When the head is making a cut, the framework of the machine is kept fixed between the roof and floor by hydraulic jacks.

Oil for the hydraulic jacks is supplied by a vane pump driven by a separate 16 hp air motor.

The lubrication chart for the K-19 machine is given in Fig. 101. The gear oil pump *4* placed in the reduction gear casing provides
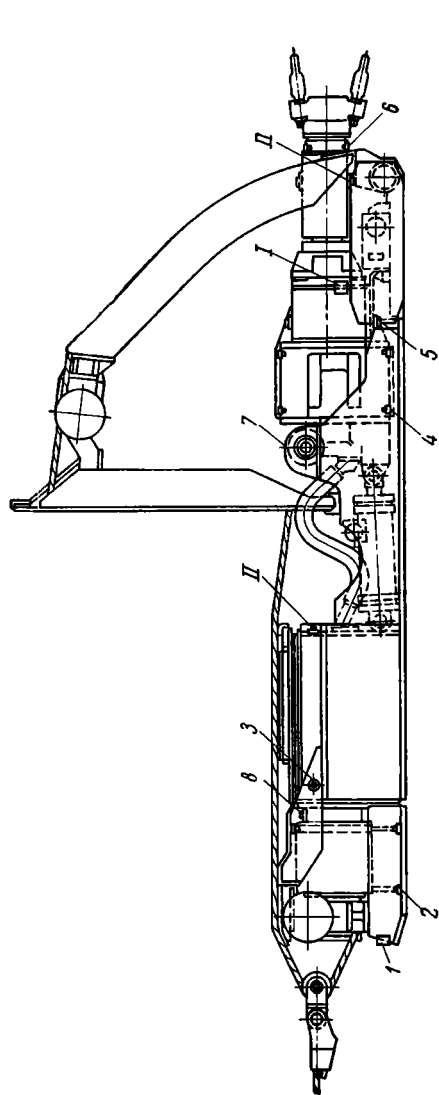
Fig. 101. Lubrication chart of K-19 coal mining machine
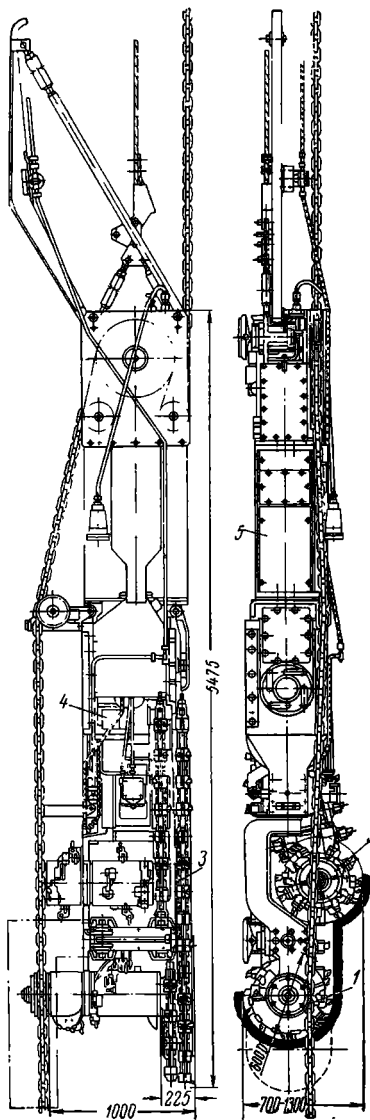
the machine with forced lubrication. The oil can be drained for re-
placement through plugged openings at points *11*. Summarized lubri-
cating instructions for the machine will be found in Table 9.

Table 9

Ref. number on Fig. 101	Number of points	Part or assembly to be lubricated	Interval	Lubricant
1	1	Hydraulic system	Oil added every 10 days	Industrial oil 20
2	4	ПШД-16 (PShD-16) air motor	Daily	УТВ (1-13) [UTV (1-13)] grease
3	2	Hydraulic system	Filled when assembled	Industrial oil 20
4	4	ПШД-28 (PShD-28) air motor	Daily	УТВ (1-13) [UTV (1-13)] grease
5	2	Reduction gear	Oil added every 10 days	Oil AK-15
6	2	Cutting unit	Daily	УТВ (1-13) [UTV (1-13)] grease
7	2	Swing shaft bearings	Every 10 days	Ditto
8	1	Pump reduction gear	Ditto	Oil AK-15

(above the table) Lubrication point

The **K-55 coal mining machine** (Fig. 102) has also been designed
to work coal of any hardness in pitching seams from 0.7 to 1.3 m
thick. The unsupported roof area is not less than 6 to 8.5 m².

The cutting head of this machine is one metre wide, and comprises
two drums *1* and *2* having a set of helically-arranged pick boxes and
picks. The two drums are driven by the cutting chain *3* fitted to cut
into the face forward of the drums, thereby steadying the machine
in operation. The upper drum can be adjusted for height by a
hydraulic device to suit the seam being worked.

This machine works downward, and has the feature of force feed
along a sized chain strung the entire length of the face.

The drums are driven by an ЭДКО4-1c (EDKO4-1s) electric motor
5 through gear box *4*.

600-1110

560

1000

УКР
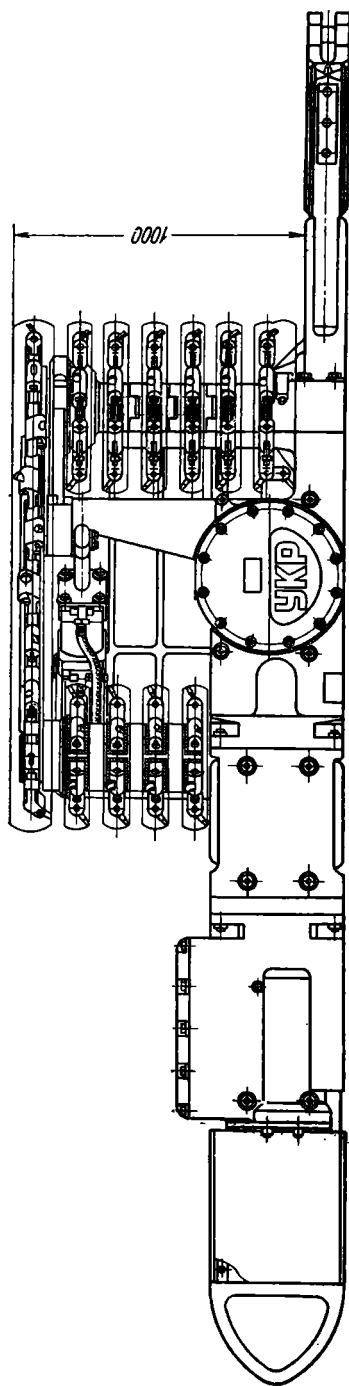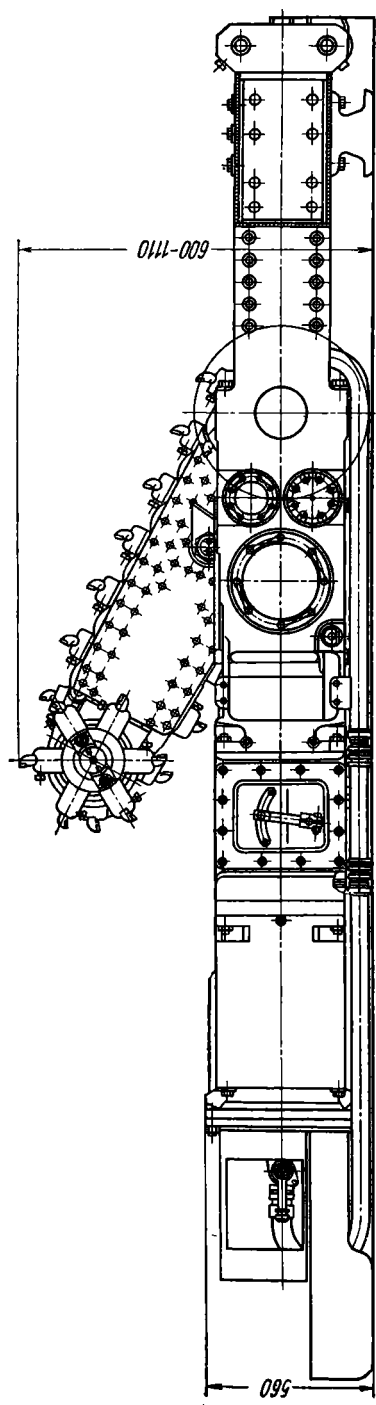
Fig. 103. УКР (UKR) coal mining machine

The **УКР (UKR) coal mining machine** is used to work coal of various hardness in pitching seams from 0.6 to 1.1 m thick.

This machine works the face off the buttock in an upward direction and requires no special roadway. The machine is hauled along the face and lowered for a new cycle by a winch mounted in an airway.

The cutting head (Fig. 103) is a rotary bar fitted with a set of cutting discs carrying replaceable picks. This bar is mounted on two supports in the reduction gear casing and is driven by chain. The face is profiled and the upper layer of coal is taken down by a cutting chain jib and a disc drum. Hydraulic jacks can raise the latter to suit the thickness of the seam being worked.

The haulage rope is attached to the machine through rigid skids ensuring stability in operation. The machine can be driven either by an electric or an air motor.

The **УВК-1 (UVK-1) system** is made up of equipment designed to mine soft and medium-hard coals in steep seams 0.3 to 0.7 m thick with a stable roof and floor.

The coal cutting unit in this installation is, in effect, a coal saw which is a frame on which 12 double pick boxes are attached. One of the picks in each box makes a cut when the saw moves upward and the other when the frame moves downward.

The coal web is broken up by shearers which widen the kerf to 300 mm. The saw can be adjusted for inclination with screw devices.

Fig. 104 gives the operating scheme of the УВК-1 (UVK-1). The saw *1* is held up against the face by the tail rope fastened to a winch *2* mounted in the airway road above. This rope passes round an upper *3* and lower *4* snatch block.

Another winch *5* lowers prop timber. When the face can be worked at sufficient rates of advance, and other conditions are favourable, the УВК-1 (UVK-1) installation can operate without having to support the face roof and without workers at the face.

КБС (KBS) coal mining system. This system is a set of machines which get and load the coal and also set up supports and control the roof.

The КБС (KBS) is intended to work steep seams 0.5 to 0.7 m thick and of 60 to 90-degree inclination, with side rock of better than medium stability. All the operations are carried out without the presence of workers at the face.

The КБС (KBS) system (Fig. 105) consists of a rapidly moving coal saw *1*, haulage winches *2*, mounted in the airway road, a metal net *3* to hold packing material, upper *4* and lower *5* rope guiding blocks, and an arrangement *6* for raising and advancing the net.

The system also incorporates water infusion equipment for dust control and loosening the coal in the seam before the saw cuts it.

7*

View A

Up to 10 m

1450 — 3250

Haulage gate

Fig. 104. Layout of УВК-1 (UVK-1) mining system

Position with
retting lifted

Section B-B'

Section A-A

Section C-C
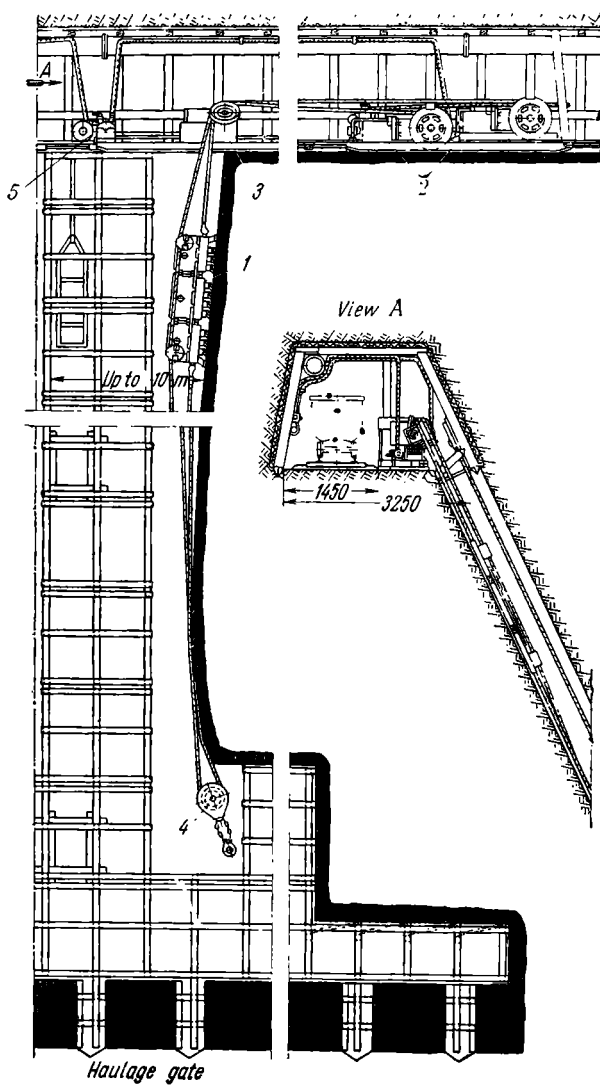
Fig. 105. Layout of the КБС (KBS) coal mining system
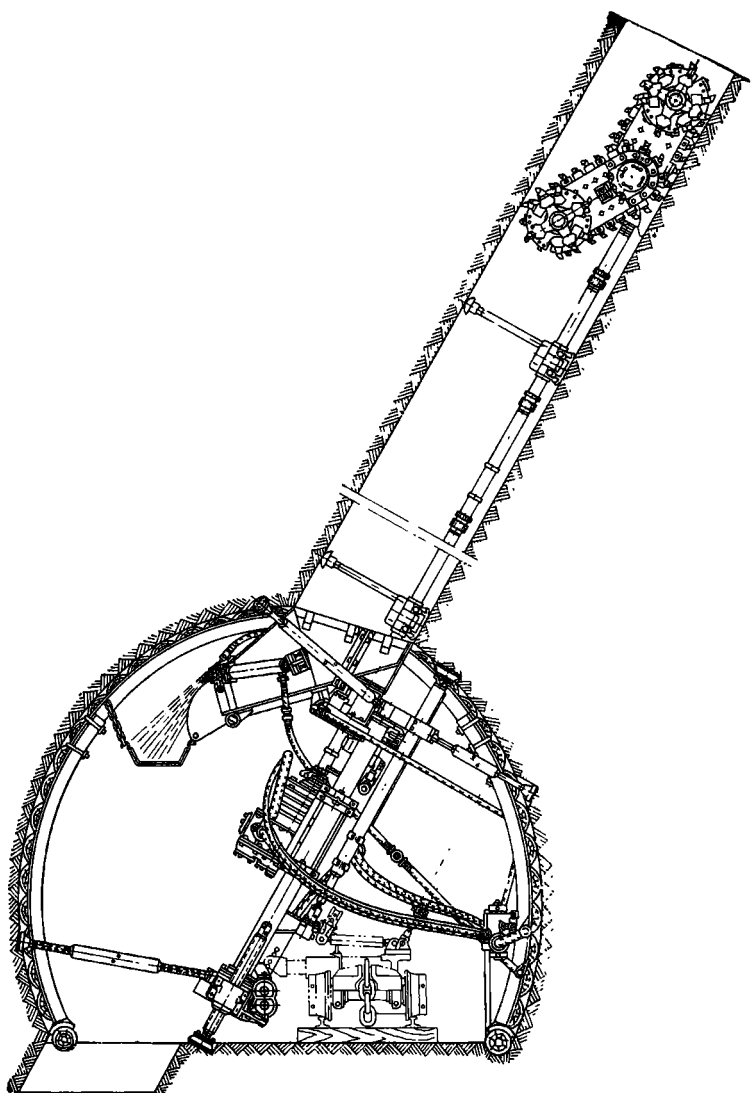
Fig. 106. КМД (KMD) coal mining system during extraction of coal

With the КБС (KBS) system, the roof is controlled by complete packing. The packing material is brought in through the airway road. Stone for packing may also be taken from the airway drivages.

The КМД (KMD) coal mining system is devised for the manless extraction of coal of various hardness from steep seams (50- to 80-degree inclination) from 0.6 to 1 metre thick and laying in medium stable strata.

The system employs the room-and-pillar method of mining, with narrow rooms cut in two steps. On the first step, a room from 1.1 to 1.3 metres wide is cut upward; during the second, or downward step, the room is broadened to 2.1 metres.

The coal-getting machine is a cutting head (Fig. 106) driven by two lines of rodding turning in opposite directions. When the room is to be widened (on the downward step), two cutting heads are fitted. The drive unit of the machine is placed in the haulage road and consists of a haulage mechanism and turning arrangement with two air motors. As the cutting head moves, a section of rodding is added on upward cutting or taken out on downward cutting.

After the rooms have been mined, the worked-out space is packed from the upper airway. The pillars are then robbed on retreat along the haulage road and very often left unpacked, while the haulage road is blocked up.

Continuous Coal Cutter-loaders for Flat Medium-height Seams

The Donbass-6 cutter-loader is a machine for working flat seams from 1.9 to 2.2 metres thick (Fig. 107).

The cutting head is driven by the same ЭДК-120 (EDK-120) electric motor as is employed in the Donbass-2k, or two МАД-191/11мр (MAD-191/11mr) electric motors (operated in tandem).

The haulage unit is the same as in the Donbass-2k.

The cutting head consists of a folding chain jib with two horizontal ripping bars set one above the other. Also, the machine has an upper ripping bar carrying cutting discs. This bar can be lowered 300 mm for flitting. With this arrangement, the coal is broken up over the entire height of the seam.

The Ukraina cutter-loader (formerly known as the Donbass-7) is intended for mining coal of various hardness in flat seams from 2.2 to 2.95 metres thick with roofs of medium stability.

It is produced in two modifications: the "low", intended for seams 2.2 to 2.55 metres thick, and the "high" for seams from 2.65 to 2.95 metres thick.

The "low" Ukraina cutter-loader has a rigid single-arm chain jib with a reach of 1.6 m and a height of 1050 mm across the picks, and
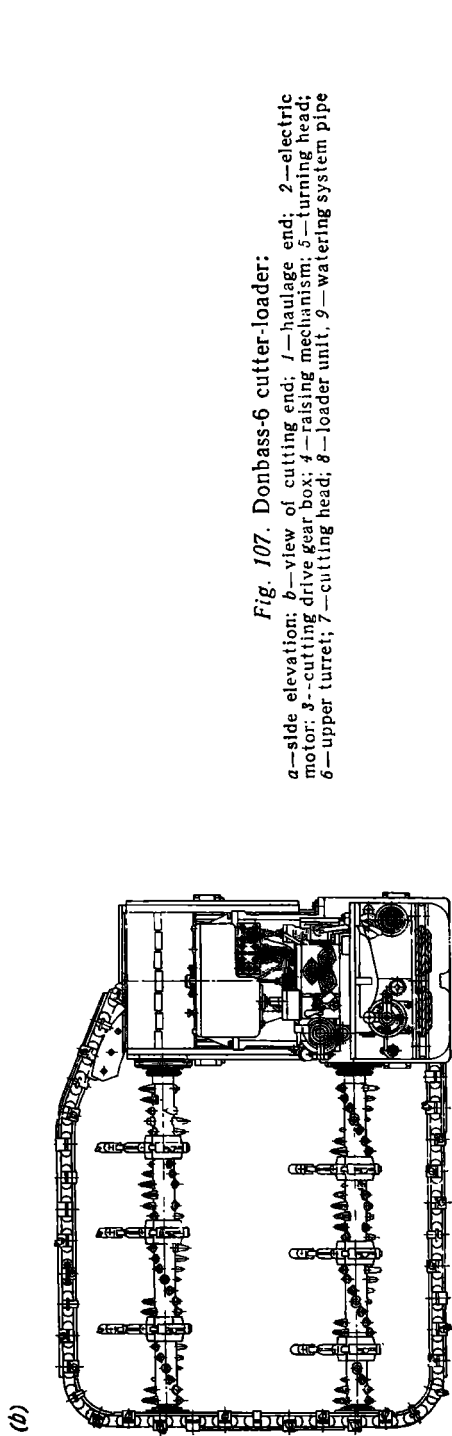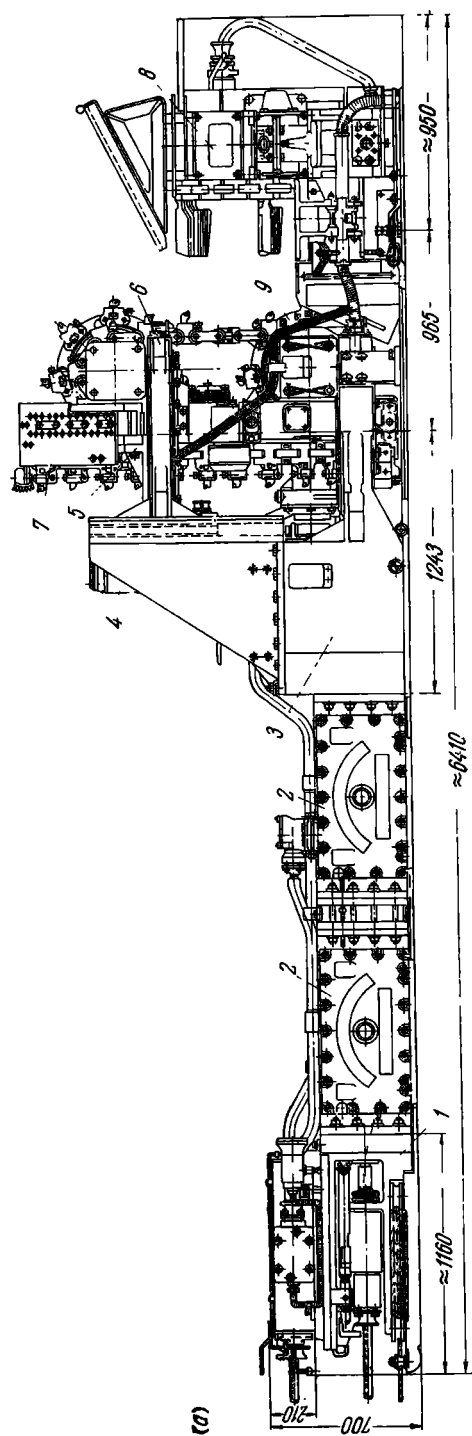
Fig. 107. Donbass-6 cutter-loader:

a—side elevation; *b*—view of cutting end; *1*—haulage end; *2*—electric motor; *3*—cutting drive gear box; *4*—raising mechanism; *5*—turning head; *6*—upper turret; *7*—cutting head; *8*—loader unit, *9*—watering system pipe
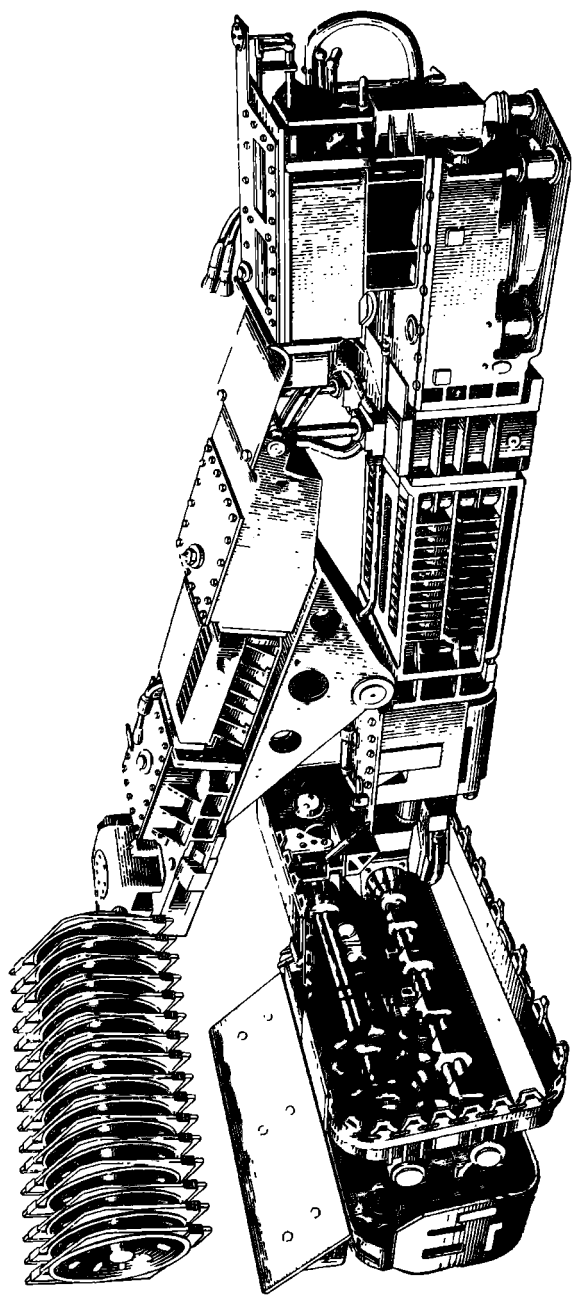
Fig. 108. Ukraina cutter-loader

an upper disc arm which can be raised or lowered during operation according to the height of the seam being worked.

The "high" Ukraina incorporates a rigid chain jib 1.5 m high across the picks, fitted with two horizontal ripping arms set one above the other. An upper disc arm completes the cutting head.

The upper disc arm (Fig. 108) is mounted at a right angle to the longitudinal axis of the machine, is driven by an individual MA-191/6к (MA-191/bk) electric motor, and can be raised and lowered by a hydraulic ram.

The haulage unit is the same as in the Donbass-2k and is driven by an ЭДК-120 (EDK-120) electric motor.

All the electrical equipment of the Ukraina operates on 660V.

The K-26м (K-26m) coal cutter-loader (Fig. 109) will extract coal of any hardness and toughness from flat seams 1.45 to 1.9 metres high and under roofs of medium stability.

The cutting head is made up of four double-pick arms *1* fixed on two vertical drive shafts extending from the gear box.

The arms rotate with their picks turning towards each other on the face side. At the same time, the arms can be made to sweep back and forth from floor to roof and thus work the full thickness of the seam.

Forward feed of the machine into the face is accomplished by hydraulic pusher rams *7* backed up by hydraulic props *6*. The sweep of the cutting head is also accomplished hydraulically by means of hydraulic actuators *2*.

When the machine cuts into the coal, it is held immovable by its two forward extensible props *3*; another prop *5* supports the roof shield *4*.

The K-26m works on a buttock down the face and cuts out a web 2 metres wide. A chain type loader *8*, with an apron jib, loads the broken coal from under the head onto a face conveyor set up directly along the face.

K-56м (K-56m) and K-57 cutter-loaders are in a class of machines adapted to extracting soft and semi-hard coals from flat seams 1.6 to 2.4 metres thick under weak roof conditions (as met with in the Moscow coal fields).

These cutter-loaders work the face off the buttock by the shuttle method and are turned round in stable holes at both ends of the face.

The K-56м (K-56m) cutter-loader has a rotary cutting head forming a truncated cone hydraulically moved in a vertical and a horizontal direction to work the coal.

In the K-57 cutter-loader (Fig. 110) the cutting head consists of a set of discs seated on a horizontal revolving rod. Hydraulic actuators sweep this head up and down the face.
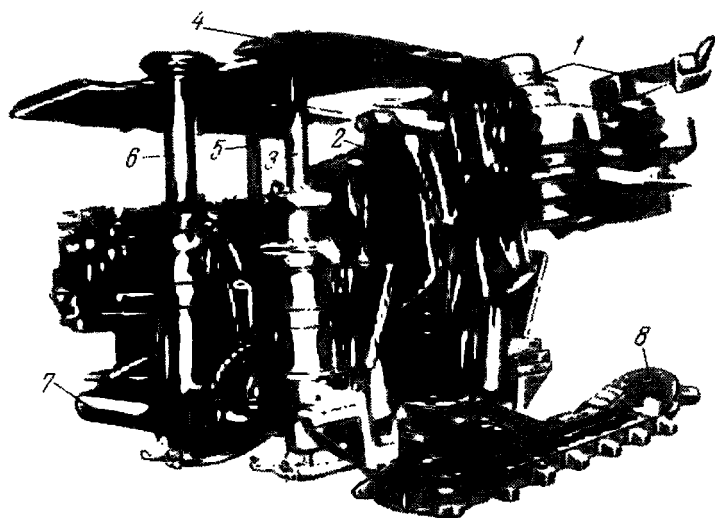
Fig. 109. K-26м (K-26m) cutter-loader
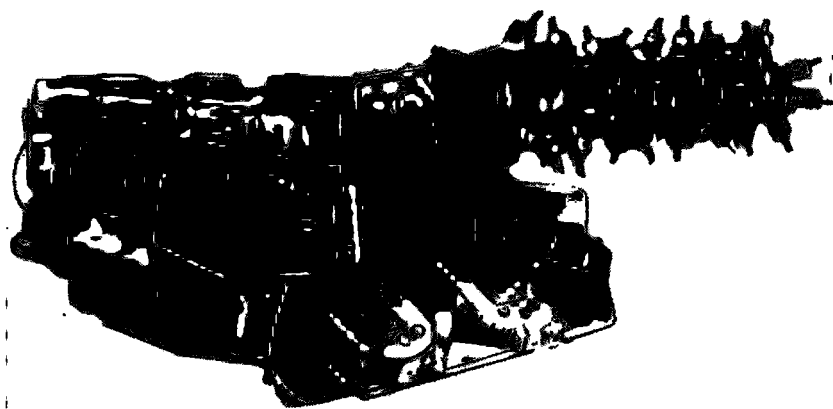
Fig. 110. K-57 cutter-loader

In both of these cutter-loaders a flat-jib loading chain gathers the coal from the face and passes it to a cross belt conveyor for transfer onto the face conveyor. Both cutter-loaders travel on crawler tracks.

The **ШK-1 (ShK-1) auger-type cutter-loader** is designed to mine semihard coal in flat seams from 1.6 to 2.4 metres thick where the roof is weak but can be left unsupported over areas of at least 2 to 2.9 square metres. This machine works the face off the buttock. The cutting head consists of picks helically fitted on a revolving auger. The cutting arm can be swung from floor to roof by hydraulic power.

The coal is loaded by worms and a pivoted mouldboard with a bulldozer attachment.

The cycle of operation with this machine is as follows. The auger cutting head is raised, the crawler bogie is started, and the cutting head is made to cut to one-half metre deep. When the overcut is completed, the crawler drive is switched off and the cutting head is lowered, bringing the web down and loading the coal onto the face conveyor. To begin the next cycle, the head is quickly raised to its upper position.

In the above discussion only longwall cutter-loaders have been dealt with. Shortwall cutter-loaders, as well as various coal-mining plant and coal ploughs will be discussed in the respective sections on integrated mining and mechanized support systems.

5-20. Safety Rules for Operation of Coalcutters and Cutter-loaders

1. In gassy and dusty mines, the methane content in the atmosphere at the face must be regularly checked as the seam is worked. Should the methane content be 2 per cent or more, all operations must be stopped and the mine inspector immediately notified. All supply of electric power must also be switched off.

2. The face roof must be maintained well supported, especially at danger points such as breaks, crevices, dome cavities, pot bottoms. The roof requires close attention and must be frequently inspected and sounded by tapping. If a hollow sound is noted, additional supports should be put in.

3. Each machine must have its frame earthed through the earth core of the trailing cable. One end of the core must be connected to the machine frame inside the cable connector shell, the other end to the magnetic starter gate-end box enclosure. Electric motor frames should be earthed through the armoured cable sheath and to a local earthing connection. Before a shift is started, the earthing connections must be examined for continuity, the terminations of the earth core checked for solid connection at the terminals in the mag-

netic starter box and at the local earthing, and the earth leakage relay examined for correct operation.

4. Where seams having inclinations greater than 25 to 30 degrees are worked, the machines should be attached to a safety rope. The winch and the snatch block for guiding the rope require fully reliable anchoring, and the rope itself must be maintained taut.

5. Trailing cables must not be used when damaged or when they have unvulcanized ("cold") splices. To avoid damage to the sheath and insulation, or electric shock, no objects should be suspended from flexible cables (for example, safety lamps). If a live cable has to be handled, it must be done only with safety rubber gloves on.

6. Before a machine can be started, it must be looked over to see that no spanners, nuts, picks, or any other objects have been left on the cutting chain jib, and that the haulage rope is correctly seated. Then the helper should be warned that the machine is to be started. The operator should never leave the controls when his machine is in operation and must always be on the alert to stop the machine immediately the need arises.

7. Utmost care must be taken when working near or on cutting chains, especially when picks are changed and cuttings are removed from around cutting jibs and heads. Carelessness may lead to the clothing being caught by a pick and end in dangerous injury.

8. Should a fire break out due to electric current, the power supply from the gate-end box must instantly be switched off and the fire put out with inert dust, sand, etc.

9. Haulage props must never be guided with the hand while the machine is running, or else the hand may be pulled into the guide roller at the drum. Wire ropes must be handled only with gloves on, or else the hands will be wounded by sticking ends of broken wires.

10. Never sit or stand near an anchor or jack prop. The machine may pull it down and an accident may result from its fall or from the fall of a pot bottom out of the roof.

11. When a machine is operating in a pitching seam, no men are allowed to be below the machine because of the danger of injury from falling lumps of coal and rock, dropped props, and even a dropped tool.

12. Only flameproof equipment shall be used in gassy and dusty mines. If a coalcutter or cutter-loader is to be flameproof, the following rules must be observed:

(a) A close fit and a tight joint must be maintained between the cover and casing of controllers, all the studs must be screwed fully home, a spring washer should be in place under every nut; after the nuts have been pulled tight, the clearance between the cover and casing should not exceed 0.1 mm. The necessary fit of the mating surfaces can be checked by feeling the heads of the three pins screwed

into the casing and ground flush with the cover top surface. If there is coal dust in the joint between the cover and casing, the pins will be sunken below the cover surface. If so, the cover must be taken off, the mating surfaces cleaned of any coal dust, and the joint re-made.

(b) The clearance between the flanges of the motor frame and haulage-unit casing should not exceed 0.1 mm; their mating faces should be clean, smooth and free from nicks over a width of at least 25 mm round the entries for electrical conductors and the controller shaft.

(c) Push button casings must be free from damage, and each must be fastened by two bolts.

(d) The mating surfaces of all other flameproof joints (between the shell and body of cable couplers, the faces of openings for push buttons and wiring fittings) must also be free from scratches, nicks and raised spots.

13. The trailing cable may be laid directly on the floor for a length of not more than 15 metres from the machine and shall be run where it will not be damaged by the machine during its work movements.

Where coalcutters and cutter-loaders work in seams up to one metre thick, trailing cables may be laid on the face floor. Live flexible cables must be laid straight and placed on supports or suspended in hanger loops.

At the end of a shift, the flexible and trailing cables should be de-energized by switching off supply at the nearest distribution centre or gate-end box.

14. When a coalcutter or cutter-loader is operated at a face with a Class III atmosphere (i.e., an atmosphere containing specified hazardous gases) or higher, a gasman must continuously be on duty at each machine.

15. No mining operations shall be carried out in dusty seams without wetting. Excellent results in dust control are obtained with the pressure water infusion method by which water is pumped under pressure into the seam through holes. The injection of water also loosens the coal in the seam and thereby raises the productivity of cutter-loaders.

E. METAL SUPPORTS

When wooden props are used for support at the working faces, the amount of timber required is large, averaging to about 40 cubic metres per 1000 tons of coal output. It is not always possible to re-use wooden props. Metal props, on the other hand, can be re-used and moved up to new positions many times. The high strength of the metal prop as compared with that of the wooden prop substantially enhances the safety of work.

When a cutter-loader advances rapidly, the prop setters are unable to put in wooden props fast enough to keep in pace. Since it is unsafe to leave large areas of the roof unsupported, the cutter-loader operator is forced to lower the haulage speed and, consequently, output of the machine.

Mechanized props and roof-support systems bridge the gap between mechanical coal-getting and prop-setting.

By function, props divide into face roof, waste-edge, and what may be termed guard supports.

Face roof props serve to support the roof along the working faces.

Waste-edge props are employed for roof control when caving or allowing the roof to subside slowly.

Guard props are set up adjacent to the face area to prevent falls of loose or ripped rock, or packing materials.

Constructionally, roof supports may be classed as separate, each taken down to the component parts (props, roof bars, yielding props and chocks); sectional, handled and set up in sections or units; and integral, also moved in sections or sets, but having component parts which are joined together to make up an integral or complete support system.

The class of integral roof supports also includes the moving shield type of supports which have a continuous form of roof shield for protecting the working area at the face from falling of rock.

Sectional supports generally consist of face-roof and yielding type prop member sections. Face-roof sections (of one or two props) joined to metal roof bars are moved forward as the cutter-loader advances, while the waste-edge yielding sections are moved to a new position by pusher-haulage units.

Complete support systems for flat and semisteep seams are usually self-propelled and employ hydraulic advance jacks. Support systems for thick pitching seams are moved under their own weight.

5-21. Metal Face-roof Props

Model M-20K, M-53K and M-53Д (M-53D) metal props differ from earlier designs in that their resistance to load rises gradually, and a greater yield is obtained.

An M-20K prop (Fig. 111) has a box-shaped body *1*, an extensible tapered member *2* with a taper angle of 3°30', a headpiece *3*, a footplate *4*, and a locking device. The locking device, housed in a body *5*, comprises a turnable horizontal wedge *6*, insert piece *7*, clamping slider *8*, return spring *9* and adjusting bolt and nut *10*. On the inside, the lock body and the slider are fitted with friction liner plates *11* attached by rivets and welding.
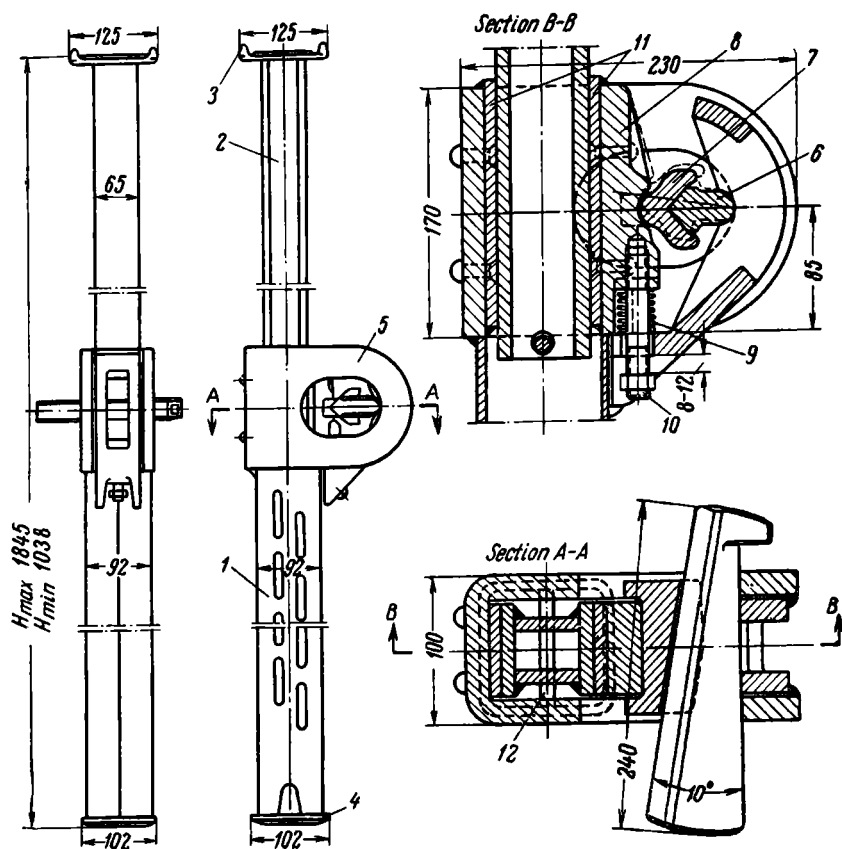
Fig. 111. M-20K metal prop

The initial set of the prop is obtained with distance wedges which can be driven into the two rows of slots in the prop body. The prop is dismantled by knocking out the check pin *12*.

At rated working load the yield distance of the M-20K prop is 300 mm. The self-tightening mechanism incorporated in the locking device begins to act after the setting load reaches 2 to 3 tons.

The M-53K prop has an extensible member tapered to an angle of 7°30′ and has a travel of 300 mm at rated load.

The high initial setting force of this prop is obtained by driving in two horizontal check wedges acting on each other through a pivoted bush.

The M-53Д (M-53D) prop differs from the M-53K prop in that it can be released remotely by means of a wire rope with an endpiece

which pulls at the arm of the holding eccentric to free the lock-re-lease wedge.

The best props are those which maintain a fixed resistance to load. They may be of the friction and the hydraulic type.

In friction props the extensible member has parallel surfaces of friction. When the load is applied to a friction prop, the extensible part yields. As the load increases the locking arrangement in the prop self-tightens, and the prop sets while maintaining a constant force of resistance.

Model TC (TS) tubular constant-resistance props are designed with a screw-type extensible part and a self-tightening mechanism in the locking device. At roof pressures less than the rated load, the prop will only sink 8 to 12 mm due to the action of the self-tightening lock.

The TC-10 (TS-10) prop (Fig. 112) consists of a tubular body *1*, tubular extensible member *2*, screw *3*, hinged headpiece *4*, footplate *5*, and a locking arrangement. The latter comprises a body *6*, clamp *7*, spring *8*, liners *9*, and horizontal wedge *10*. The initial jacking action is obtained by turning out the screw *3* by means of bar arms *11*.

The TC-25 (TS-25) prop (Fig. 113) has a box-shaped body *1*, a tubular extensible part *2* to which a headpiece *3* and hinged roof plate *4* are welded, and a locking arrangement.

The locking arrangement consists of a body *5*, clamp *6*, spring ring *7*, liners *8* and a horizontal drive-in wedge *9*.

The two rows of slots in the prop body receive check wedges driven into them when the prop is initially jacked up.

Model ЛC (LS) constant-resistance props are distinguished by the fact that the extensible member is kept restrained by a metal band so arranged as to provide six friction surfaces in the prop locking device.

The ГC (GS) constant-resistance hydraulic props comprise a cylindrical body and tubular extensible member acting as a plunger or ram. An oil reservoir within the prop supplies the working fluid. The extensible member incorporates a hydraulic piston pump, a yield valve, and a release valve. The hydraulic pump is operated by a handle, and serves to build up the jacking pressure in setting up the prop. The function of the yield valve is to maintain the resistance of the prop at its rated yield value. The release valve relieves the pressure on the ram when the prop must be moved to a new place.

ВКШ-2 (VKSh-2) temporary walking supports are designed for roof control above the loader of a Donbass cutter-loader where seams from 1.6 to 3 metres high are being worked under unstable roof conditions. These supports consist of two units connected together by a truck frame permitting the released unit to be moved along
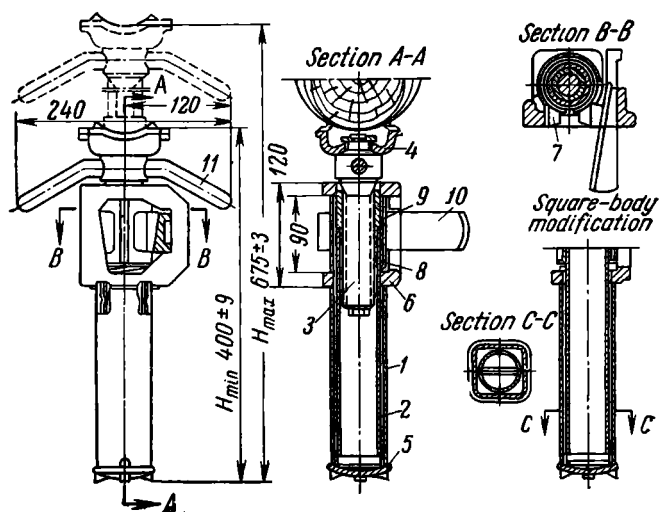
Fig. 112. TC-10 (TS-10) metal prop
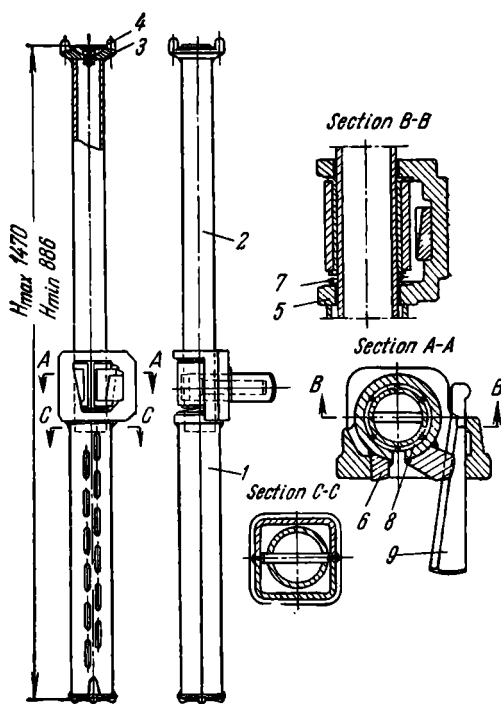
Fig. 113. TC-25 (TS-25) metal prop

the set unit. Each unit makes use of tubular type TC-10 (TS-10)
props in which the yielding locking arrangement has been replaced
by a rigid bar lock. The roof is supported by wooden bars the ends
of which are placed on permanent support props. ВКШ-2 (VKSh-2)
temporary supports find application in the Moscow coal fields. They
are obtainable in four sizes for seams 1.45 to 2.15; 1.8 to 2.5; 2.3 to
2.9; and 2.65 to 3.3 metres high. Their weight is, respectively, 130,
145, 165 and 170 kg.

5-22. Metal Roof Bars

The purpose of metal roof bars is to provide a cantilever form of
roof support above a machine road and control the roof exposed as
the seam is worked.

M-31K hinged roof bars (Fig. 114) are cast beams *1* with corrugated
walls. One end *2* of the bar is the shape of a fork, the other end *3*
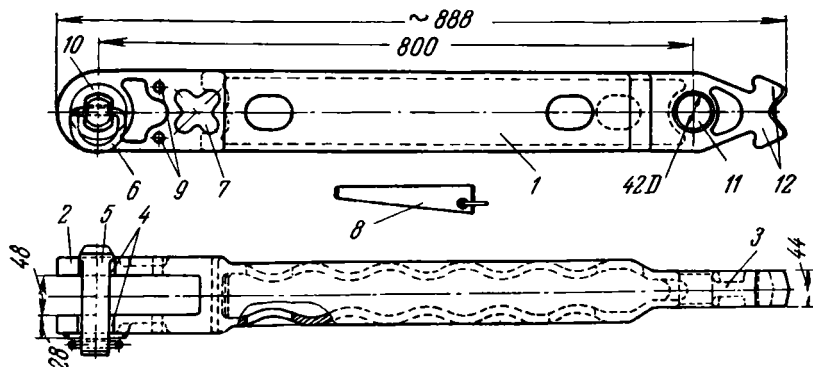
Fig. 114. M-31K metal hinged roof bar

has an eye. The hole *4* receives a pivot pin *5* carrying a ring *6*. The
cross-shaped hole *7* takes a removable locking wedge *8*, while the
two holes *9* take a locating pin. The ring *10* prevents pin *5* from falling
out. The eye end has a hole *11* for a pin and support lugs *12*.

M-45K cast hinged roof bars differ in that they have a wedge-
shaped jointing pin and a locating wedge which extends above the
highest part of the bar beam.

CB (SV) welded hinged roof bars do not differ from M-45K bars,
except that they are of welded construction, lighter in weight and
lower in strength.

5-23. Waste-edge Supports

Waste-edge props and chocks are set up in line where the roof is to be caved and are designed to prevent the roof from subsiding beyond the selected goaf area during and after caving.

The ОКУ (OKU) waste-edge yielding prop or chock (Fig. 115) comprises a cap *1*, which has a domed uppersurface and a spherical thrust bearing undersurface and pin at the bottom; a self-braking
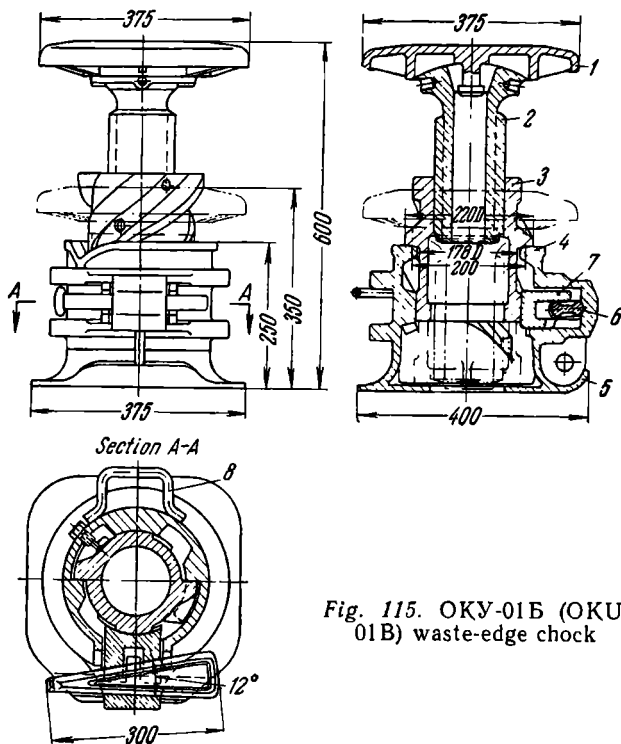
Fig. 115. ОКУ-01Б (OKU-01B) waste-edge chock

screw *2* for providing a larger height range and for setting the prop, and an extensible member *3* with a self-braking thread on the inner surface, holes for a tommy bar, and four helix threads, two thrust, and two braking, on the outer surface. The cast hollow base *4* gradually widens in the baseplate *5*.

This chock also has a cast drive-in wedge *6*, a brake shoe *7* held pressed against the braking helix of the extensible member and a welded carrying handle *8*.

In the ОКУ (OKU) chock the yield under load is obtained due to the elastic deformation of the locking device.

In modified ОКУ (OKU) chocks the extensible part has a triple-thread instead of a double-thread screw. In this way the stability of the extensible member in the body has been improved.

The ОКД (OKD) chock is simi-lar to the ОКУ (OKU), but its locking arrangement is different and uses a vertically-swung lever arm having an eccentric head against which the wedge presses to maintain the friction block in con-tact with the conical thread of the extensible member. When the chock is remotely released, the eccentric-head arm is pulled through an angle to free the friction block and allow the extensible part to retract into the chock body.

OKM chocks differ from ОКУ (OKU) chocks in that they incor-porate an eccentric-lever type lock-ing mechanism permitting of re-mote release.

M-85 chocks feature a locking arrangement using a double wedge shifted by an unequal-arm lever. The lower end of this lever is held in the locked position by a journal pin with a release handle. A turn of this handle is all that is needed to release the chock.

M-32 chocks (Fig. 116) are of the constant-resistance type and also have an arrangement for remote release. They consist of a cap *1*, screw stack *2*, ring nut *3*, barrel *4*, pedestal *5*, and a band brake. The cap has a spherical bearing on the undersurface with a pin welded in the centre. To receive the pin and support the cap, the screw stack

Fig. 116. M-32 waste-edge chock

ends in a spherical surface with a hole in it. The locking arrangement in this chock consists of a nut *3* and a band brake embracing the nut round its outside surface.

The special features of the M-32 are a large yield distance and the

use of a brake band passed round the nut and maintained tight by a set of springs through an angular lever. When the load on the chock rises to its yield value, the nut rotates and allows the screw stack to retract into the barrel.

F. INTEGRATED COAL MINING SYSTEMS, MECHANIZED SUPPORT SYSTEMS

An integrated coal mining system consists of machines and related equipment providing for full mechanization of all the main mining operations at the face, i.e., the getting, loading and face transport of the coal, conveyor advance, roof control, and setting of roof supports.

Integrated coal mining systems may be divided into those for wide-web extraction (with a 1.2 to 2 m working width of the cutting unit), those for short-web extraction (with a 0.4 to 1 m working width of the cutting unit), and those built around a coal plough unit.

5-24. Mechanized Supports and Coal Mining Systems for Wide-web Coal Extraction

The M-9 integrated mining system is intended to work flat seams 0.55 to 0.75 m thick, and comprises a type УKT-2м (UKT-2m) or КЦT (KTsT) cutter-loader, a КС-15 (KS-15) portable scraper-chain conveyor, an M-9 sectional support system and an M-10 pusher-haulage unit equipped with a mechanical drive winch.

The M-9 sectional support system (Fig. 117) is made up of face roof and waste-edge sets, called chocks.

A face-roof unit consists of two TC-10 (TS-10) props *1* and a flexible roof bar *2*. A waste-edge chock set consists of a type OKM or M-85 chock *3*, a TC-10 (TS-10) prop and a flexible roof bar hinged to the chock.

The waste-edge chock set is moved forward by the M-10 pusher-haulage unit comprising an electric-driven double-drum winch with a KOФ-11-4 (KOF-11-4) motor, and four vertical and one horizontal hydraulic rams. One of the drums is used to haul the waste-edge chock set by rope, the other for haulage of the pusher-haulage unit itself up and down the face.

The vertical rams anchor the pusher-haulage unit in the work position between the roof and the floor. The horizontal ram is used to advance the conveyor. The hydraulic system of the pusher-haulage unit incorporates a pump driven through the reduction gear of the winch.

As the cutter-loader advances along the face, the face-roof units of the M-9 support system are moved by hand and set up so that the fore
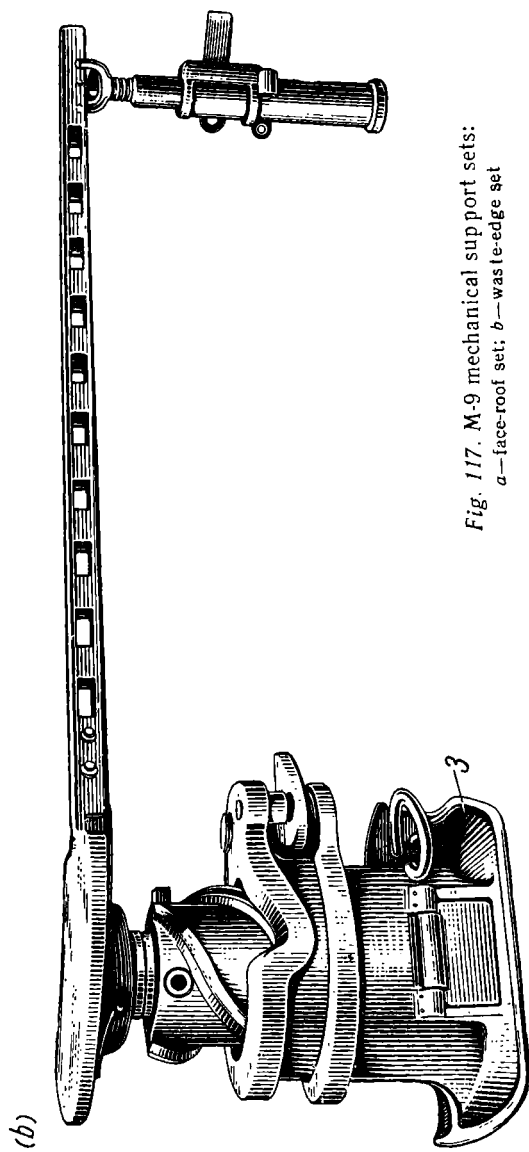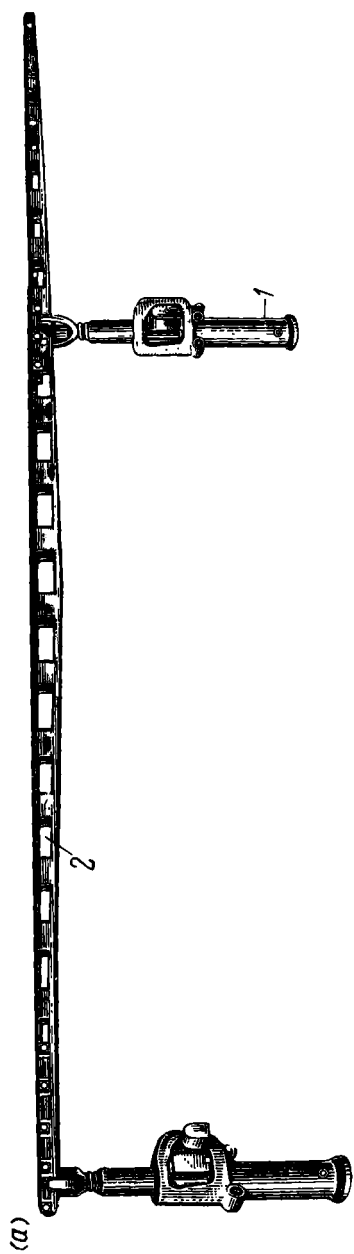
Fig. 117. M-9 mechanical support sets:
a—face-roof set; *b*—waste-edge set

prop is at the face and the aft prop is beyond the conveyor. A corridor 2.2 m wide is thus formed beyond the cutter-loader to permit the conveyor to be moved forward by the pusher-haulage unit. After an advance of the conveyor, the waste-edge chock sets can be moved forward. To do so, the front end of the roof bar is first lowered to the floor, the TC-10 (TS-10) prop is taken out, and the whole unit is then pulled up by rope to the new position with the M-10 winch drum.

After the cutter-loader has removed a web of coal from the face, it is turned round in a stable hole at the face end to begin the next cut in the opposite direction.

The М-9Д (M-9D) integrated system is used to work seams 0.9 to 1.4 m high and comprises a Donbass-1 [or some other wide-web cutter-loader such as the ЛГД-1 (LGD-1), Gornyak-1, Kirovets, etc.], an М-9Д (M-9D) sectional support system, a snaking КС-9 (KS-9) or КС-15 (KS-15) scraper-chain conveyor, and an М-10Д (M-10D) pusher-haulage unit.

The М-9Д (M-9D) support system uses TC-15 (TS-15) props, M-85 chocks, and high-strength weld-fabricated roof bars.

Fig. 118 shows the layout of the M-9D integral mining system at a face when operating with an ЛГД-1 (LGD-1) cutter-loader.

The integral mining system incorporating an M-77 support system is intended to work flat seams 1 to 1.6 m thick and consists of an M-77 sectional hydraulic support system, a Donbass-2k or ЛГД-2 (LGD-2) cutter-loader, a КС-9 (KS-9) snaking conveyor, and an M-82 pusher-haulage unit.

In this system the support sections and the face conveyor are moved up to the face by the pusher-haulage unit while the cutter-loader advances and is 10-15 metres forward of them. After the web of coal has been extracted and all the support sections and the conveyor have been moved up the entire length of the face, the pusher-haulage unit and the cutter-loader can be hauled into the face-end stable hole.

The M-82 pusher-haulage unit is placed beyond the conveyor and can move the support sections whether working the face from gate end or to gate end.

The M-77 hydraulic support system is a development which has grown out of experience gained with the МПК (MPK) rigid support system. It consists of single-prop units which are extensible ГС (GS) hydraulic props connected to lengthened roof bars supported on the face side by the prop and by a bracket on the roof plate of the hydraulic waste-edge chock on the goaf side.

The roof plate of the waste-edge chock is fitted with a piston-spring pump which feeds the oil from the reservoir into the hydraulic system of the chock when the latter is set up. When the section is released, the oil flows from the hydraulic system back into the oil
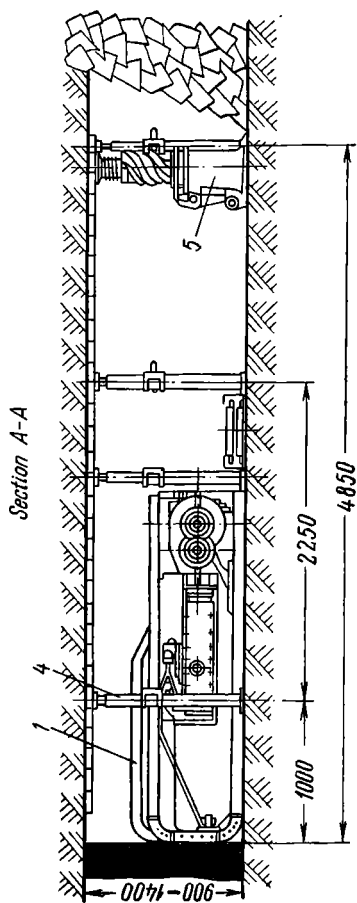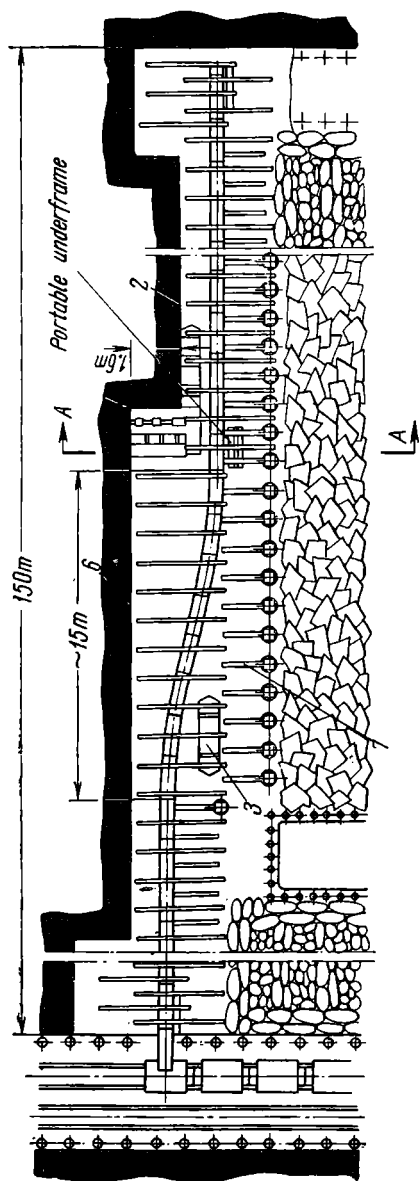
Fig. 118. Layout of M-9D mining system operating with an ЛГД-1 (LGD-1) cutter-loader:

1—cutter-loader; 2—snaking scraper-chain conveyor; 3—M-10D pusher-haulage unit; 4—TC-15 (TS-15) face-roof metal prop; 5—waste-edge chock; 6—face-roof bar; 7—waste-edge bar

reservoir in the chock base. When the chock is jacked up, the M-82 pusher-haulage unit presses with its plate on the projecting dome of the pump piston after the roof plate has come into contact with the roof, and the pressure on the chock builds up.

The screw part incorporated in the extensible strut of the chock doubles the extension obtained by hydraulic means to 750 mm.

The M-77A support system incorporates a pneumatic accumulator in the hydraulic system. It automatically pushes the piston outward when the hydraulic pressure is released. Such an arrangement gives a residual jacking force of 1 to 1.5 tons and permits the section to be moved up without loss of contact with the roof. The flexible leaf-spring roof bars in this support system readily adapt themselves to the undulations of the roof.

5-25. Mechanized Support Systems and Coal Mining Systems for Narrow-web Coal Extraction

Narrow-web extraction is characterized not only by the limited cutting depth or reach of the cutter-loader (0.4 to 1.0 m) and the rapid rate of advance during cutting (2.5 to 3 m/min), but also by substantial differences in the method of working the seam owing, above all, to the absence of props between the conveyor and the face, the employment of cantilever roof supports, and the move-up of the face conveyor fully assembled while the cutter-loader continues to extract the coal.

Originally, narrow-web extraction was carried out with one of the conventional cutter-loaders fitted with a shorter cutting head and loader. Later, however, specially designed cutter-loaders were developed. They can operate in shuttle (advancing and retreating) fashion and are equipped to extract the roof web of coal.

The Integrated Coal-mining System Incorporating a K-52м (K-52m) Cutter-loader is designed for mining coal and anthracite of any hardness and toughness in flat seams from 1.1 to 1.7 metres thick.

The equipment used in this system is a K-52м (K-52m) narrow-web cutter-loader (Fig. 119) which is hauled along the frame of a KC-9 (KS-9) or KСП-1 (KSP-1) snaking face conveyor, and a set of metal props, hinged roof bars and waste-edge chocks..

The cutting unit of the K-52м (K-52m) cutter-loader comprises, a lower *1* and an upper *2* ripper drum, both of which carry a set of picks. As the coal is broken down, it is loaded onto the conveyor by the picks and pick boxes which throw the coal onto a mouldboard *4* placed alongside the lower drum.

The diameter of the lower drum is 930 mm, and of the upper drum 670 mm. Cutting widths of 0.5; 0.625 or 0.75 m can be obtained by fitting the lower drum with additional rings carrying pick boxes.

Dimensions visible on figure: 1100-1600, 480 in flitting position, 1500, 2325, 900, 4840, 1070, 1290, 500-750

Labels: 2, 3, РВ, ГП41, K52M
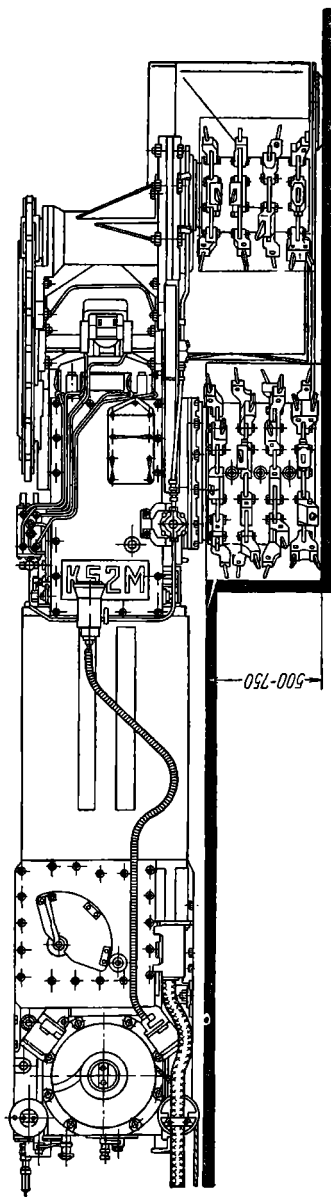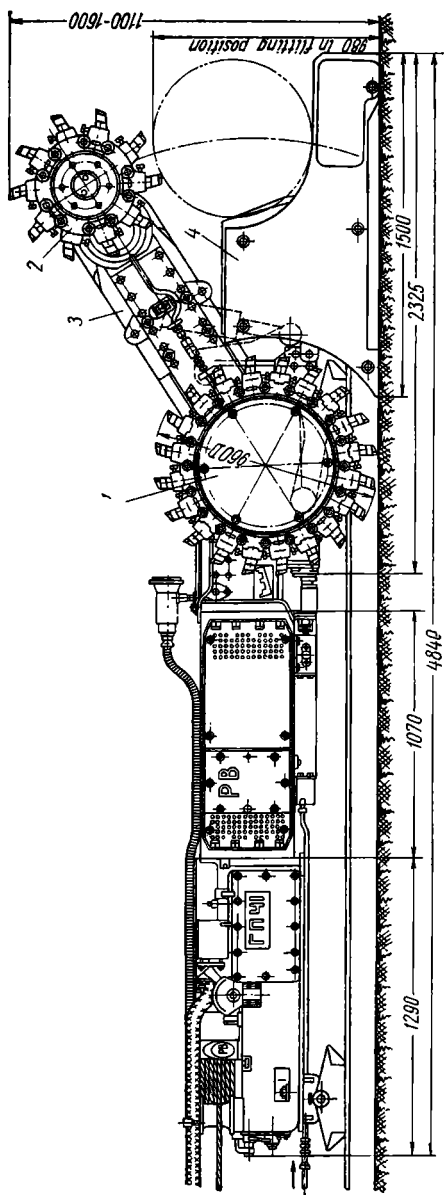
Fig. 119. K-52м (K-52m) cutter-loader

Frame *3*, carries the upper drum so that it can be turned about the axis of the lower drum by two hydraulic rams. This permits the cutter-loader to cope with varying seam thickness and extract the coal from floor to roof.

The haulage unit in the K-52м (K-52m) is the type ГПЧ-1 (GPCh-1) employing hydraulic power.
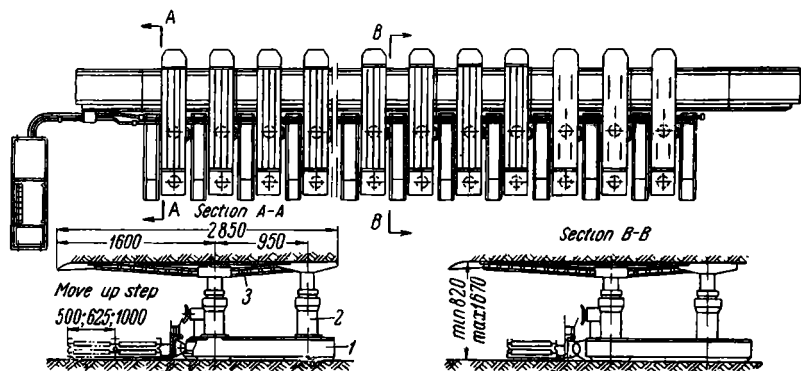
Fig. 120. M-87 hydraulic-powered support

This cutter-loader works the face from gate-end up. On the return run its mouldboard trims the face and loads the remaining coal onto the conveyor. After the machine reaches the gate end, the road is cleaned up and the conveyor simultaneously moved over.

Operation of the K-52м (K-52m) cutter-loader in conjunction with the M-87 hydraulic-powered support is also a practical combination.

The M-87 support system has been designed to support and control roofs in flat seams from 0.95 to 1.6 metres high and lying between roofs and floors of medium stability.

The M-87 support system (Fig. 120) uses the same type of units both for the face-roof and the waste-edge sections. The common base for this support system is a KC-9 (KS-9) scraper-chain conveyor. Support units are set up along the face on 0.95 m centres. Each unit consists of a cast base *1* with spherical pivots carrying two hydraulic props *2*, each having a load resisting capacity of 75 tons. At the top, the props hinge with flexible roof bars *3* made up of two leaf-spring elements. One of the latter points in the direction of the face so that its cantilever end supports the roof above the conveyor.

The hydraulic advance ram is built into the base and is hinged to the conveyor frame. Depending on the cutting width of the cutter-loader head, the support system can be advanced in steps from 0.5 to 1 metre.

A leaf spring attached to the side of the base bears upon a guide beam made fast to the trough framework of the conveyor section. This beam fixes the direction of advance of the support system.

The support units can be advanced either one after every other or in some other sequence.

The integrated system incorporating the M-87 support system is designated as the КМ-87 system. Instead of a К-52м (K-52m) cutter-loader, it can use some other cutter-loader capable of haulage along the frame of the face conveyor [the УМК (UMK), УДК (UDK)].

The integrated mining system incorporating a КУ-2 (KU-2) narrow-web cutter-loader is designed to extract coal of any hardness and toughness from seams 0.8 to 1.25 thick with roofs of at least medium stability.

This system includes a КУ-2 (KU-2) cutter-loader, a KC-17 (KS-17) snaking scraper-chain conveyor, metal props, hinged roof bars and waste-edge chocks.

The КУ-2 (KU-2) cutter-loader works the face from the buttock in shuttle fashion. Stable holes at the face ends need not be prepared in advance, since the machine will cut them itself.

The КУ-2 (KU-2) system is shown in Fig. 121. Cutter-loader *1* has its cutting unit placed between the face and the conveyor *2*, and its own gear box on the other side of the conveyor. It therefore operates from the floor and from behind the conveyor, and as it advances, moves forward the conveyor and travels on the conveyor roadway with its skid bearing upon the props of the waste-edge chocks *5*. The hinged roof bars *4*, jacked up against the roof by metal props *3*, support the exposed roof at their cantilever ends. The cutting head of the cutter-loader has both a slant drum carrying a set of cutting picks and a vertical telescopic cutting drum to break down the upper coal web. Cutting height can be varied at will by raising and lowering the upper drum.

The broken coal is loaded onto the conveyor by the cutting picks and the gathering arms attached to the horizontal drum.

Each end of the machine has two cutting heads. They cut on a buttock one metre wide. Haulage is by rope and the haulage unit is hydraulic-powered.

The КУ-2 (KU-2) cutter-loader has several advantages over many other cutter-loaders, namely:

(A) The operator runs the machine from beyond the conveyor and under a supported roof area.

(B) The unsupported roof area is kept to a minimum.

(C) It can cut its own stable holes and work in shuttle fashion, thus eliminating the loss of time on the advanced preparation of stable holes and on the idle return in one-way operation.
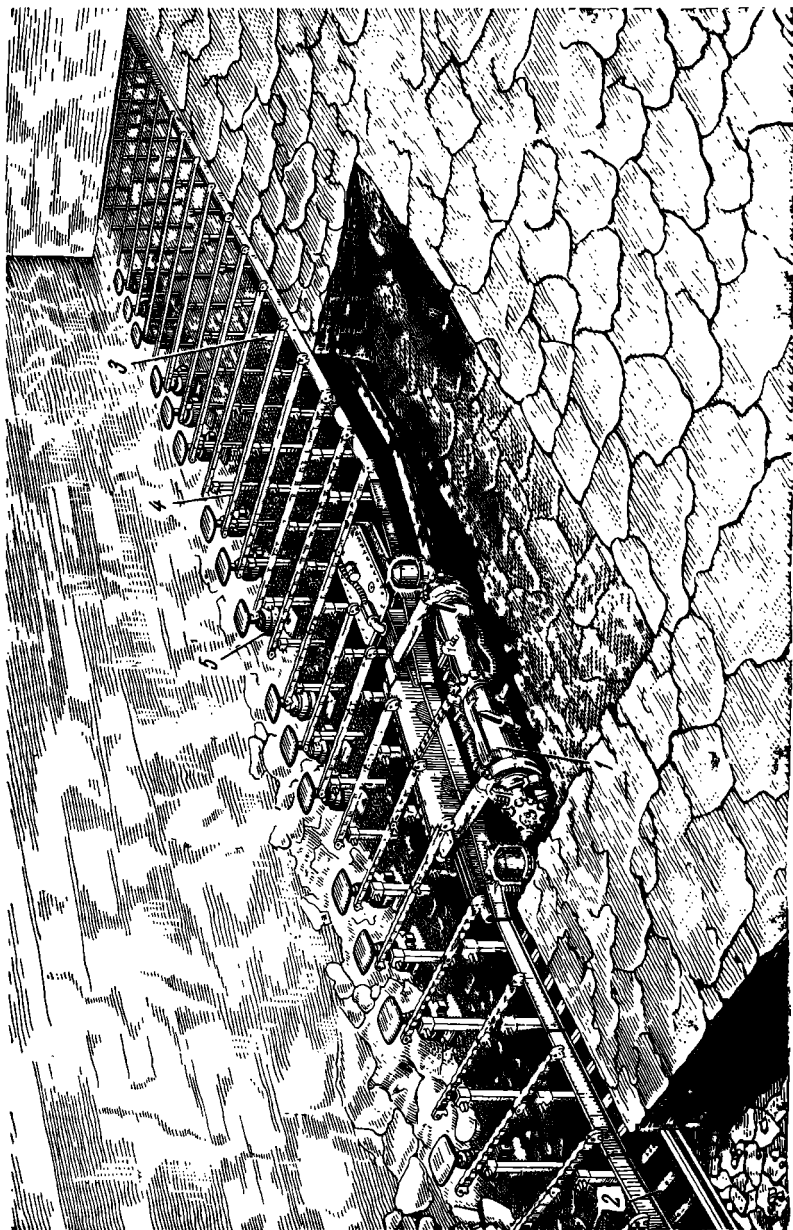
Fig. 121. Mining system with a КУ-2 (KU-2) cutter-loader

(D) The conveyor is moved over by the cutter-loader, and a pusher unit may be dispensed with.

(E) The trailing cable requires no protection as it is laid beyond the conveyor along the second half of the roadway.

A shortcoming of the KУ-2 (KU-2) system is that the roof control is not mechanized. At present a combination of the KУ-2 (KU-2) cutter-loader and the M-97 mechanized support system is being tried out.

The integrated coal mining system incorporating the УДК-1 (UDK-1) cutter-loader is designed to work coal of any hardness and toughness in flat seams from 0.9 to 1.25 metres high and consists of the УДК-1 (UDK-1) short-web cutter-loader, a KC-9 (KS-9) snaking scraper-chain conveyor, metal props, hinged roof bars and waste-edge chocks.

The УДК-1 (UDK-1) cutter-loader works the face off the buttock. It makes a cut one metre wide, and is driven by an ЭДК-120 (EDK-120) electric motor. The trailing cable is brought in through the side wall of the motor frame.

The cutter-loader 2 is symmetrical in design (Fig. 122), having a cutting head and a loader on each end. Owing to this, the machine operates in shuttle without having to turn at the ends of the face.

Each of the motor shaft extensions is fitted with a claw clutch for connection to the right-hand and left-hand gear boxes through which the respective cutting heads and loaders are driven.

Each of the reduction gears incorporates a rotary frame whose purpose is to support the respective cutting head. Rigidly attached to it is a wormwheel sector which meshes with a worm mounted so that it is vertical in relation to the gear casing. With the aid of this worm the cutting head can be adjusted for height to suit the seam thickness.

The worm gearing is actuated by a ratchet mechanism mounted in the gear box and connected to an eccentric seated on the loader drive sprocket shaft.

The ratchet operates continuously, but it actuates the worm only when the jumper plate is shifted to the right or left. Then, the cutting head can rise to work a seam 0.72 to 1.25 metres thick, or sink 30 mm below the machine footing for removal of the bottom coal.

Each cutting head is a rod rotating at an angle of 12 degrees relative to the buttock face and fitted with a set of cutting discs separated by spacer sleeves. The rod shaft rides in two bearings. On the face side, the rod shaft carries a drive sprocket whose end face is studded with cutting picks.

Motion from the reduction gear is transmitted to the cutting head by the cutting chain. In order to maintain tension in the chain,
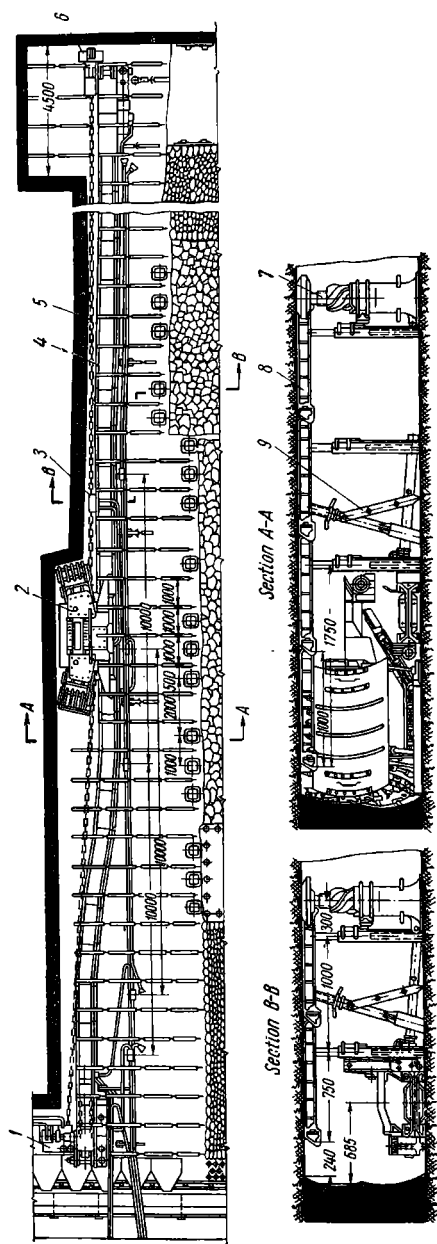
Fig. 122. Coal mining system with УДК-1 (UDK-1) cutter-loader

the support nearest to the face carries another sprocket also studded with cutting picks.

In operation, the leading head makes a cut in the bottom part of the seam and the trailing cutting head takes care of the remaining leaf of coal.

Both loaders, of identical construction, operate simultaneously; one loading the coal coming from the top web, and the other the coal from the bottom web. Each loader is of cast bar design with guide ways bolted to the side walls of the bar to carry the double-hinge loading chain.

Where the coal is transferred on to the conveyor *4*, the tail-end of the loader is set at an angle of 30 degrees. Its flat end is supported by a ramp permanently attached to it. Hinged to this ramp is another hinged ramp, connected by rods to the cutting head, and kept pressed against the floor. Consequently, when the cutting head cuts into the bottom coal, the loader transfers it onto the conveyor. When the cutting head is working the top coal, the hinged ramp swings through 58 degrees to chute the coal to the loader.

Two guard panels, one on each side, protect the men working at the face from flying coal thrown outward by the loader. These panels have troughs for the electric cables and water supply hoses. They also carry ФВУ-1к (FVU-1k) flameproof lighting fittings supplied by the 660/36V lighting transformer mounted in the electrical assembly box.

The cutter-loader as a whole is supported and guided by two skids, each in the form of a steel tube with shoe plates welded to it. One skid tube is used to pass the haulage chain through and secure it.

Cutter-loader movement is effected by a sized chain *5* and a haulage winch *1* placed in the stable at the gate end and anchored by two hydraulic props which have self-contained hydraulic systems and can be set by operating the oil pump with an electric hand drill.

A friction-clutch pulsating control mechanism is incorporated in the haulage winch. The disc cam on the pulsator shaft bears upon a roller which, in turn, causes the pulsator rod to reciprocate. Its motion is transmitted by a follower gear and tie rod to the friction clutch, whose discs are alternately compressed and released.

Haulage speed can be adjusted from zero to 2.5 m/min by setting the cam-gear remotely or by hand. Tractive pull can reach 15 tons.

The remote speed control mechanism consists of an АОЛ-0.12 (AOL-0.12) motor of 80W, and a speed reducer giving a speed of 1.2 rpm at the follower gear. It is housed in a separate flameproof chamber in the winch gear case.

Tension in the haulage chain is set up by an arrangement *6* anchored at the far end of the face by two jacking props. This arrangement has a racking jack similar in construction to the props.

The trailing cables and water supply hoses are laid and moved mechanically by a cable carriage *3* which rides on the conveyor.

Running the entire length of the conveyor, on the goaf side, there is a cable trough. The trailing cables and hoses are bunched together and permanently fixed in the trough bottom as far as the mid-point along the face. At this point the cables are brought out from the bottom of the trough, run round the cable winch drum and into the electrical assembly box so that they can be laid or taken from the top of the trough.

Prior to starting work, when the cutter-loader is still in one of the stable holes, the cable carriage is positioned midway along the face. As the cutter-loader advances, the cable carriage is also moved by the haulage chain, the upper and lower strands of which are passed through a shaped guide groove in the frame and over the upper and lower sprockets of the carriage reduction gear. Both sprockets engage through an intermediate gear and are in a train with a gear ratio of 1 : 3.

The cable carriage, therefore, travels at half the speed of the cutter-loader, either taking up the cable or paying it out down into the trough.

The ЭДК-120 (EDK-120) electric motor of the cutter-loader is controlled by two ПМВИ-1365 (PMVI-1365) magnetic starters. They have intrinsically-safe control circuits and are connected for reversing duty.

An АП660/127в (AP660/127V) starting unit with reversing contactors controls the АОЛ-0.12 (AOL-0.12) motor of the remote haulage speed control mechanism and also supplies the lighting circuit of the face loading point.

The haulage-winch motor is remotely controlled by means of a ПМВР-1451 (PMVR-1451) reversing magnetic starter box.

A ПМВ-1331 (PMV-1331) magnetic starter box automatically switches on the K-11-4, 660-V electric motor of the watering system.

In an emergency, the ЭДК-120 (EDK-120) drive motor can be switched off by the AP-120 (AR-120) emergency isolating switch.

At the beginning of a cycle, the cutter-loader is in the gate-end stable hole and the КС-9 (KS-9) conveyor is laid out in a straight line along the face. The roof of the conveyor road is supported by the cantilever ends of the bars set on the props behind the conveyor.

As the cutter-loader advances alongside the conveyor with a skid bearing against it, the conveyor is moved to the face at a distance of 20 metres behind it by jacks *9* actuated by an electric hand drill.

Hinged roof bars *8* are put in to support the exposed roof as soon as a web of coal has been cut off the face. In the meantime the type ОКУ (OKU) waste-edge chocks *7* are moved forward in the head end

of the face. At the end of a cut, the cutter-loader cuts a stable hole
at the head end for a second web to be made on its retreat. In doing
so, the cutter-loader travels alongside the conveyor, by now fully
moved up in one line along the face. While the cutter-loader is remov-
ing the second web, the chocks are moved forward along the gate end
portion.

The advantages offered by the УДК-1 (UDK-1) system are:

(1) Shuttle operation can be carried out;

(2) The conveyor is snaked over while the cutter-loader continues
to extract the coal;

(3) Some of the roof control (advance of waste-edge chocks) work
is also done while the cutter-loader cuts coal;

(4) No time is lost on shifting the anchor props; the laying out and
taking up of the trailing cable and water hose is mechanized;

(5) No time is lost on getting the top coal web down;

(6) Stable holes are cut mechanically.

Experience acquired in pilot operation of the УДК-1 (UDK-1)
cutter-loader has led to the development of a more advanced modi-
fication, designated the УДК-2 (UDK-2).

The MK-1 multicutter-loader system mines coal of any hardness
in flat seams 0.7 to 1.0 m thick and employs several MK-1 cutter-
loaders which work the face simultaneously, each over a given
section.

The cutting head on the cutter-loader is a vertical pick drum. Hy-
draulic rams can extend the drum as much as may be necessary for
cutting a given seam. The machine is hauled at the face, and the
drum is driven, by a calibrated chain powered from drives placed at
the ends of the face. The loading is done by a double-hinged cutting
chain also transmitting motion to the cutting drum from the reduction
gear.

Stable holes are cut by the cutter-loader itself while the conveyor
is advanced by hydraulic jacks. The props are moved forward by
stationary jacks operated by a central pump set up in the gate road.
These stationary hydraulic jacks or rams are spaced 20 to 25 metres
apart along the face and attached to the conveyor frame.

Each cutter-loader works within its section in shuttle fashion. It
cuts a web 0.8 m wide from the face on each trip.

Four cutter-loaders are required to work a face 150 metres long.

**The integrated mining system using the УМК-1 (UMK-1) narrow-
web cutter-loader** works flat seams 1.1 to 1.5 metres high and consists
of a УМК-1 (UMK-1) cutter-loader, a КС-9 (KS-9) snaking conveyor,
ТС-25 (TS-25) or ГС (GS) metal props, metal hinged roof bars, waste-
edge chocks, a conveyor-advance unit, and a chock-shifting unit.

The УМК-1 (UMK-1) cutter-loader (Fig. 123) works on the shuttle
principle and uses the scraper-chain conveyor frame as its track.

8*
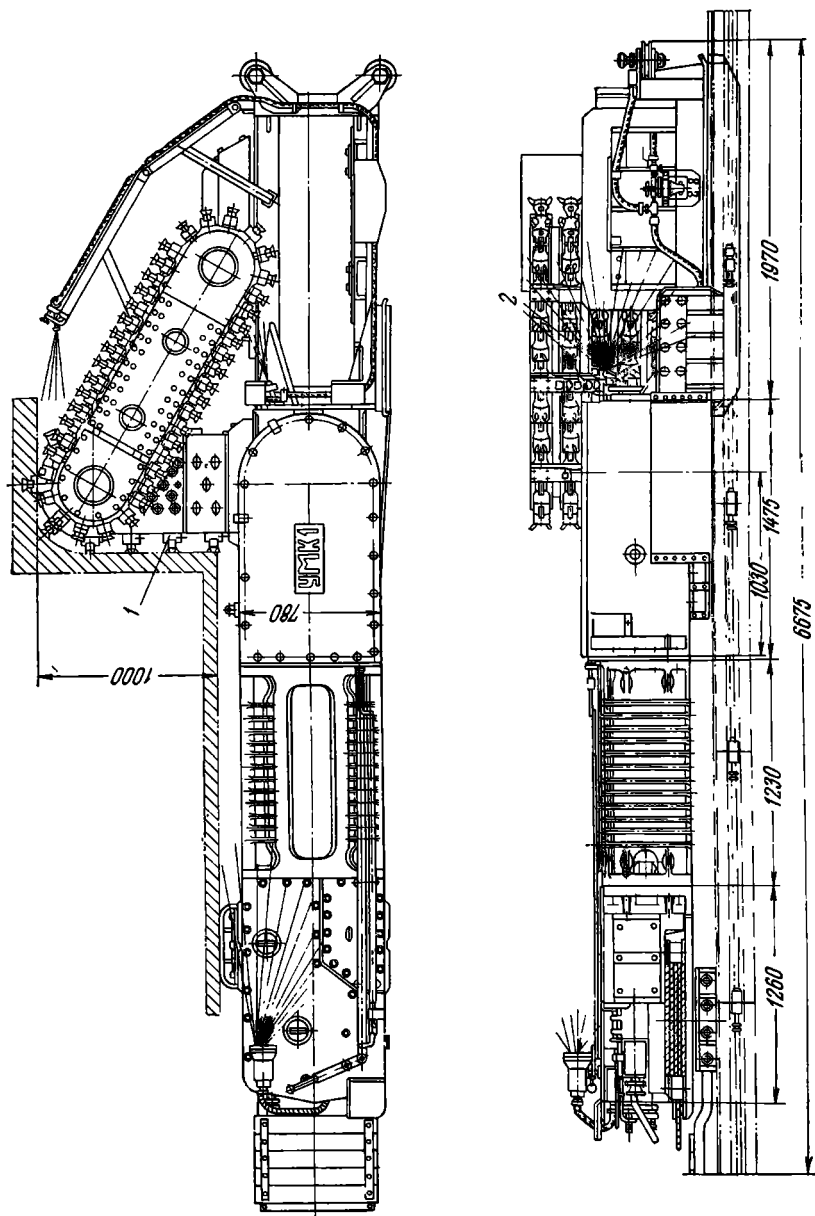
Fig. 123. УМК-1 (UMK-1) cutter-loader

Its cutting end has a stepped jib *1* set at a right angle to the frame of the machine. The double-hinged cutting chain of this jib cuts a kerf 190 mm high and simultaneously drives a set of five small flat jibs *2* arranged at an angle of 30 degrees with the frame of the machine. These small jibs carry cutting chains fitted with picks having carbide-tipped bits which cut both in a forward or reversed direction.
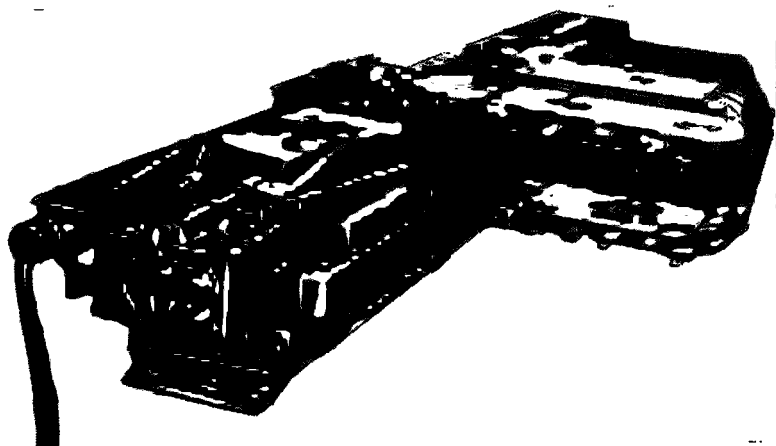
Fig. 124. КРД (KRD) cutter-loader

Within the lower part of the flat-jib stack two hydraulic rams are furnished to raise or lower the upper jib in accordance with the height of the seam to be extracted.

The УМК-1 (UMK-1) cutter-loader has been tried out in the Proletarskaya-Glubokaya Colliery in the Donbass field at a face 160 metres long with an average output of 404 tons per day and a face output of 6.6 tons per man-shift. The monthly face advance was 34.2 metres. One shortcoming of this cutter-loader is that small coal accounts for more than 40 per cent.

The КРД (KRD) narrow-web cutter-loader works medium-hard to hard but fractured and friable coals from seams 0.7 to 1.1 m thick with roofs of less than medium stability. It works on the shuttle principle and uses the same road as the face conveyor.

The cutting head of the cutter-loader (Fig. 124) has a loop and a flat jib. Two chains move in the guides of the loop jib; the forward one cuts and the rear chain loads the coal. Before the machine is started in the opposite direction, three cutting picks and three loading flights are interchanged on the chains.

The flat jib is mounted on a hydraulic column which can raise or lower the jib to suit the height of the seam to be worked.

A feature making for more stable two-way operation of the cutter-loader is that the reduction gear of the cutting head and the cutting head itself are arranged in the middle of the machine, between the electric motor and the haulage unit.

When the cutter-loader travels with the haulage end leading, the flat jib is ahead of the loop jib and makes an overcut in the seam. Then the loop jib cuts the coal down. As lumps of coal come out of the loop jib, the loading chain breaks it up and loads through openings in the cutting assembly frame onto the face conveyor. On a return run, when the machine moves with the electric motor leading, the flat jib makes the overcut behind the loop jib. Loading is facilitated by gathering aprons.

As the cutter-loader extracts a narrow web from 1 to 1.4 m wide, the snaking conveyor is advanced without disassembly, and the exposed roof is controlled by hinged metal bars supported along the waste edge by OKУ (OKU) chocks and along the face side by metal props.

5-26. Mechanized Support Systems and Mining Systems for Narrow-web Extraction of Medium-height Seams

The Mosbass mechanized support system is intended for faces such as are encountered in the Moscow coal-fields and are of the support-shield or hydraulic shield type, a class used to support a narrow strip of the face and permit the roof to subside on its roof shield.

The Mosbass support may be used in flat undulating seams from 1.8 to 2.5 metres thick. At a face of 50 to 60 m length, from 62 to 75 support sections will be used.

Coal at faces worked with the aid of Mosbass supports is transported by a KC-2 (KS-2) scraper-chain conveyor set up along the full length of the face on the bases of the support sections .

Mosbass supports are built as individual sections 800 mm wide hinged to each other at their bases along the full face length so that the supports can follow the undulations of the floor.

Each section (Fig. 125) consists of a base *1*, carriage *2*, shield *3*, cap piece *4*, hydraulic ram *5*, hydraulic drive unit *6*, working fluid distributor *7*, hood *8* and hydraulic system hoses. In the base two guideways are furnished for carriage *2*, and two brackets are fitted to receive the trunnions of hydraulic drive unit *6*. On the face side the base has a platform for the conveyor sections. At the bottom the shield *3* is joined to the carriage, at the top it is attached to the cap piece *4*. A skirting of 8 mm steel plate partially encloses the shield on its sides.

The cap piece *4* is hinged to the shield on the caving side and to the hydraulic ram *5* in the middle. The hydraulic drive unit *6* is a double-acting hydraulic jack or ram.

Oil for the hydraulic support system is supplied under a pressure of 200 atmospheres by a power pack pump of 50 litres per minute capacity stationed in the gate road.
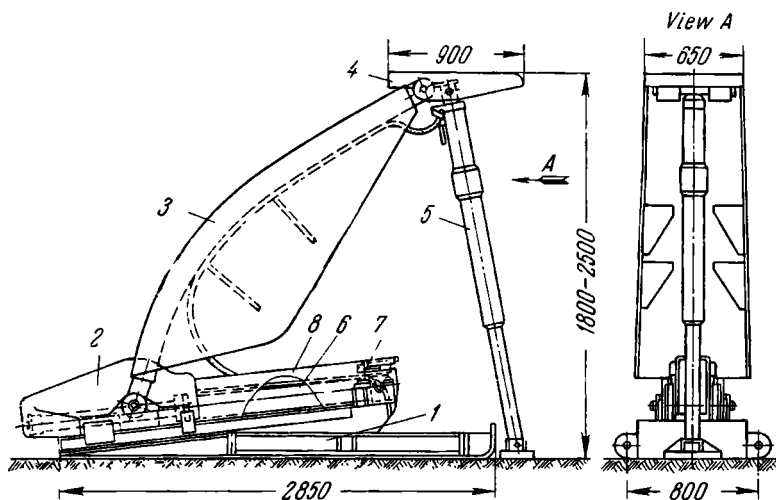
Fig. 125. Mosbass hydraulic-shield support section

The oil from the pump is delivered to the distributor from which it is directed into the pressure oil line or the base advance line.

Coal is extracted under Mosbass supports by drilling and shot firing.

The supports are moved forward as follows.

After the face has been cleared, the distributor is switched to feed oil to the piston space in the support hydraulic drive unit and the pump is started. The piston rods of the hydraulic unit then bear upon the standing carriage while its cylinder end pushes at the brackets in the base. This causes the base to move towards the face together with the conveyor section it carries. When the base and conveyor advance is completed, the shot holes drilled beforehand are fired and the loading of the coal is begun. As this work proceeds, the cap pieces and shields, and the carriages are moved further up, one section after another. Before the cap piece can be moved, the hydraulic ram or prop must be unloaded to permit its lower and attached footpiece to be pulled forward, this being done by feeding pressure

oil to the rod part of the ram. In doing this, the lower end of the re-
leased hydraulic prop is slanted towards the face and the oil is ad-
mitted into the rod part of the hydraulic drive unit to move the car-
riage, shield and cap piece forward. When the cap piece has been
moved over, the hydraulic prop is jacked up between the floor and
the cap by feeding the pressure oil to the piston space of the prop.

While the cap piece is being brought forward, a temporary prop
is put in to support the roof.

Mosbass-3 powered shield supports have been successfully operat-
ed in the Moscow District coal field and have given high outputs
per man per shift. A material disadvantage of this support is its
inability to operate in conjunction with a cutter-loader because of
the hydraulic prop which slants in the direction of the face.

The experience gained in the operation of the Mosbass-3 supports
has made possible the development of new designs of powered hy-
draulic shield-support systems capable of working in conjunction with
cutter-loaders [such as the Щ-57 (Shch-57), AK, and OMK supports].

The Щ-57 (Shch-57) system comprises Щ-57 (Shch-57) supports, a
scraper-chain conveyor and a Ural-2c (Ural-2s) or BM-1 (VM-1) nar-
row-web cutter-loader.

The Ural-2s cutter-loader (Fig. 126) will extract coal of any hard-
ness from flat seams 1.5 to 2.5 m thick under difficult geological con-
ditions where large numbers of boulders are encountered and the roof
is weak and not more than 2 square metres of roof area can be left
unsupported.

It can be operated in conjunction with both Щ-57 (Shch-57) sup-
ports and other pieces of equipment; in particular, it has been suc-
cessfully used in the Vorkuta coal mines in combination with a
snaking scraper-chain conveyor and individual supports.

The track for the Ural-2s cutter-loader is the frame of the scraper-
chain conveyor. The cutting head is a spiral-type jib carrying a set
of cutting picks.

No stable holes need be prepared in advance. The spiral jib, turned
along the axis of the machine in flitting, is turned by a hydraulic
mechanism to make a starting cut when the machine is at the gate
end.

This cutter-loader operates in cycles. At the beginning of a cycle,
the cutter-loader is advanced through 0.7 to 0.8 m with the cutting
head lowered to floor level, and the head slices off the bottom web
of coal. Then the haulage drive is switched off, and the head is raised
to break down the upper web, while loading some coal in the process.
When the head reaches the roof, it is lowered, and its rotation helps
to load the coal onto the conveyor. The remaining coal is loaded on
the conveyor by the loading plough arranged behind the cutting
head. This completes the work cycle, and the next cycle begins.
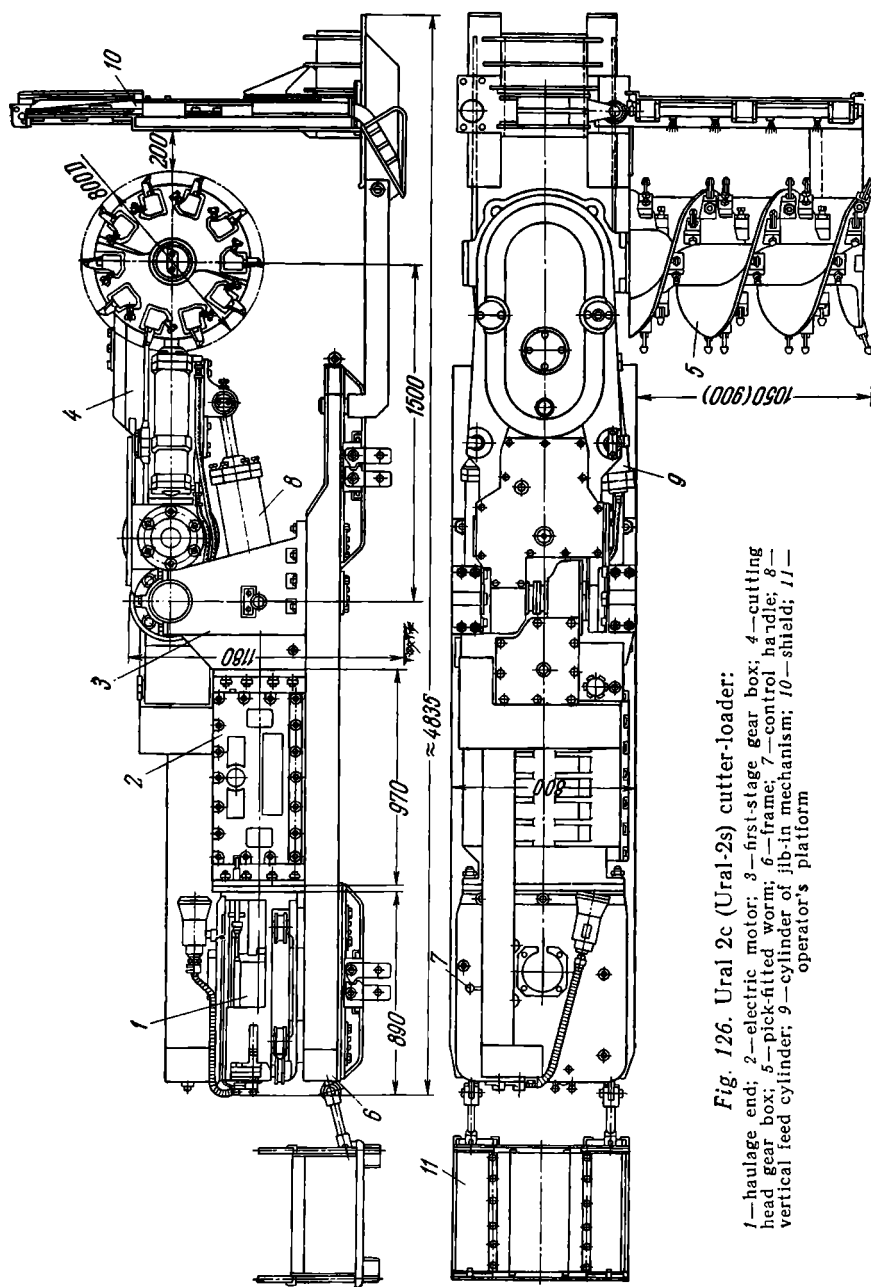
Fig. 126. Ural 2c (Ural-2s) cutter-loader:

1—haulage end; *2*—electric motor; *3*—first-stage gear box; *4*—cutting head gear box; *5*—pick-fitted worm; *6*—frame; *7*—control handle; *8*— vertical feed cylinder; *9*—cylinder of jib-in mechanism; *10*—shield; *11*— operator's platform

The Ural-2s cutter-loader has also been successfully used in the Chelyabinsk coal fields where cutter-loaders cannot usually work because of boulders. The principle on which the Ural-2s works is such that the machine can easily skip past the boulders.

The BM-1 (VM-1) coal cutter-loader, designed for shuttle operation, uses the conveyor frame as its track and has a revolving-bar cutting-loading head. This machine has been scheduled for test operation in conjunction with Щ-57 (Shch-57) supports.
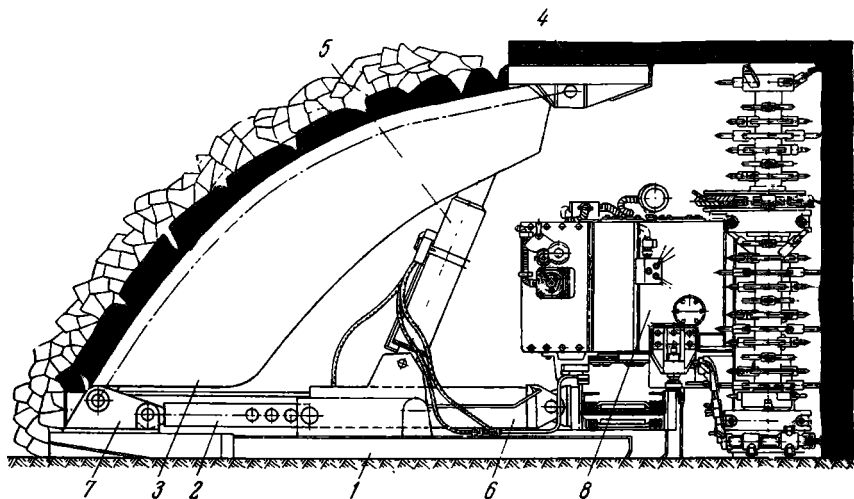
Fig. 127. Щ-57 (Shch-57) shield-support sections used in conjunction with BM-1 (VM-1) cutter-loader

An understanding of the main features of the BM-1 (VM-1) machine may be gained from Fig. 127 in which may be seen the Щ-57 (Shch-57) support comprising a base *1*, tie bar *2*, shield *3*, cap piece *4*, hydraulic jack prop *5*, horizontal hydraulic moving jack *6* and a carriage *7*, and the BM-1 (VM-1) cutter-loader *8*.

The support base, shield and hydraulic jack prop are hinged together. The cap piece is likewise connected to the shield by a hinge joint. The connecting rod of the horizontal moving jack is connected to the scraper-chain conveyor base structure, while the piston cylinder end is attached to a carriage connected by stiff tie rods to the fixed support bracket of the hydraulic jack prop.

After the cutting-loading machine has passed by, the sections are brought up against the conveyor by a pull of the moving jack, one at a time and with the hydraulic jack prop released.

After the web of coal has been extracted and all the support sections have been pulled up to the conveyor and jacked, the horizontal

jacks will bear upon the support sections and be actuated simultaneously to advance the conveyor as a whole the full length of the face.

The **AK system** consists of AK supports, a КСП-1м (KSP-1m) conveyor and a K-58 cutter-loader.

The K-58 cutter-loader works in shuttle fashion from end to end of the face, with the conveyor serving as its roadway. One under-support rides upon the built-up sideboard of the conveyor, and the other two under-supports on skids welded to the conveyor trough.

The cutting units of the K-58 cutter-loader are three drums fitted with picks; the two lower drums are placed one at each end of the machine, the third drum occupies an upper position. The latter is adjustable in height and can be set in accordance with the thickness of the seam to be worked.

Haulage of the cutter-loader along the face is accomplished by a sized-link chain stretched from end to end of the face, and a hydraulic haulage unit. As the coal is broken down by the drum picks, it falls on a mouldboard and, from it, onto the conveyor. The remaining coal is loaded on the conveyor by a ploughshare which follows the second trailing drum.

The K-58 cutter-loader can be operated in conjunction with both AK and M-81 supports (see the M-81 system described a little further).

AK supports (Fig. 128) are constructed as individual sections joined together by means of underframes. Each section has an underframe *1*, skids *2*, shield *3*, cap piece *4*, two hydraulic jack props *5*, and a hydraulic moving ram *6* (the K-58 cutter-loader is at *7* in Fig. 128).

Being horizontally hinged to each other, the underframes form a common bed, rigid in the horizontal plane, but permitting of partial vertical articulation. This common bed mounts the face conveyor. A skid in an opening of the frame serves as the footing for the hydraulic jack props which carry the cap pieces of the support section. This skid is joined to the underframe by the horizontal hydraulic moving ram. The cutter-loader advances along the underframes on ways provided on the frames. Each support section, and the face conveyor as a whole, are moved forward in the same way as when operating with Щ-57 (Shch-57) supports.

When the conveyor is brought forward, the cutter-loader drums cut their way into the coal face, and so the work can proceed without driving stable holes at the ends of the face.

The two hydraulic jack props incorporated in each support section make for high stability during operation of the cutter-loader, but lower the manoeuvrability to some degree.

The **OMK system** consists of a set of OMK supports, a scraper-chain conveyor and a КУ-60 (KU-60) cutter-loader. The latter is a narrow-web cutting machine (taking a web 0.5 m wide) designed for single-
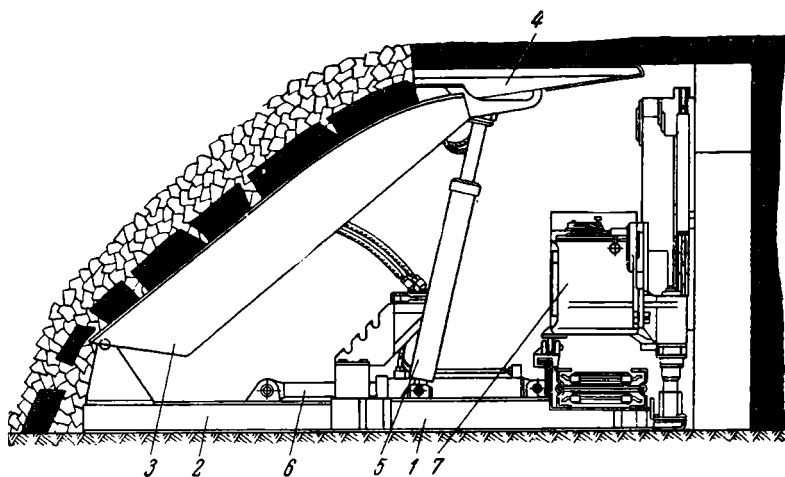
Fig. 128. AK shield support
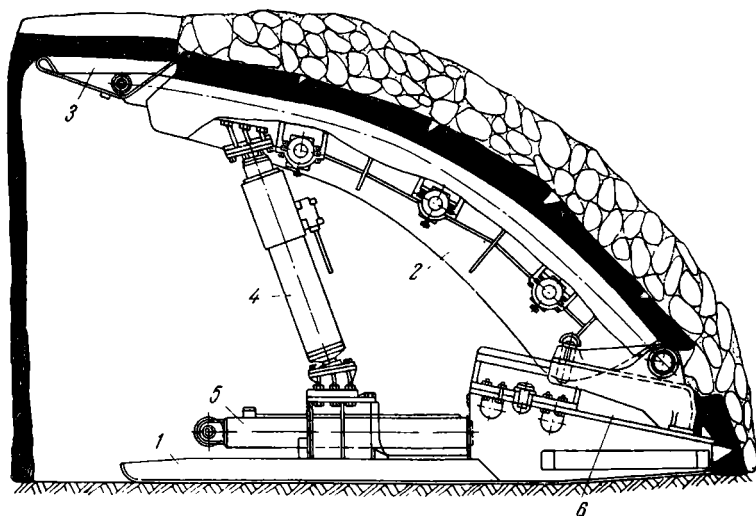
Fig. 129. OMK shield support

direction operation. It comprises a ГПЧ-1 (GPCh-1) hydraulic haulage unit with a rope drum, an MAO-60 electric motor with a 60-minute rating of 80kW, a lower cutting unit of the drum type with a reduction gear and loading ploughshare, and an upper cutting head incorporating a drum fitted with picks and a shearing jib. The cutting units may be raised and lowered hydraulically. The track for this cutter-loader is the scraper-chain conveyor frame.

OMK supports are in the form of separate sections one metre wide and are joined to each other by the conveyor frame structure. Each section (Fig. 129) has a base *1*, shield *2*, cap piece *3*, hydraulic jack prop *4*, horizontal hydraulic ram *5* and a carriage *6*. The inter-section gaps are bridged by movable hoods.

The scraper-chain conveyor in the OMK system is sectional, each section being one metre long, with a trough 588 mm wide and 70 mm deep. The sections are hinged so that the cutter-loader can follow the undulations of the floor.

As the cutter-loader advances along the face, the support sections are pulled up closer to the face one after another. When the web of coal has been extracted the full length of the face, the cutter-loader is returned with the aid of the conveyor chain. Then the whole conveyor and the cutter-loader seated on its frame, are pushed up to the face for starting the next cycle.

Tests of the OMK system in the No. 2 Zubovskaya pit have given good results. On some days up to 800 tons of coal was cut at a face 50 m long. This works out to a section output of 30 tons per man-shift and a face output of 50 tons per man-shift. The average monthly face advance was 63 metres.

The system using M-81 supports is designed for mining coal in flat seams from 1.5 to 3.2 metres thick and comprises a set of M-81 hydraulic shield supports, a K-58 or КУ-57 (KU-57) coal cutter-loader and a conveyor.

The КУ-57 (KU-57) cutter-loader (Fig. 130) is designed to extract soft and medium-hard coals and requires an unsupported area of 4 to 5 square metres for its operation.

The cutting unit on this cutter-loader is a loop chain jib dimensioned to cut a thin web of 0.95 m height which is broken up as the jib advances along the face. A special feature of the КУ-57 (KU-57) machine is the narrow body (450 mm wide), achieved by "on-edge" design of the machine. The КУ-57 (KU-57) cutter-loader is, in fact, an improved modification of the BOM-53 (VOM-53) machine.

Operation with the cutter-loader is carried out on a shuttle cycle without any need for idle trips. The haulage unit incorporated in the machine is the same as used in the КМП-3 (KMP-3) coalcutter. On a cutting run, the rope passes through a pulley block and the machine can thereby travel at a speed ranging from zero to 0.7 m/min.
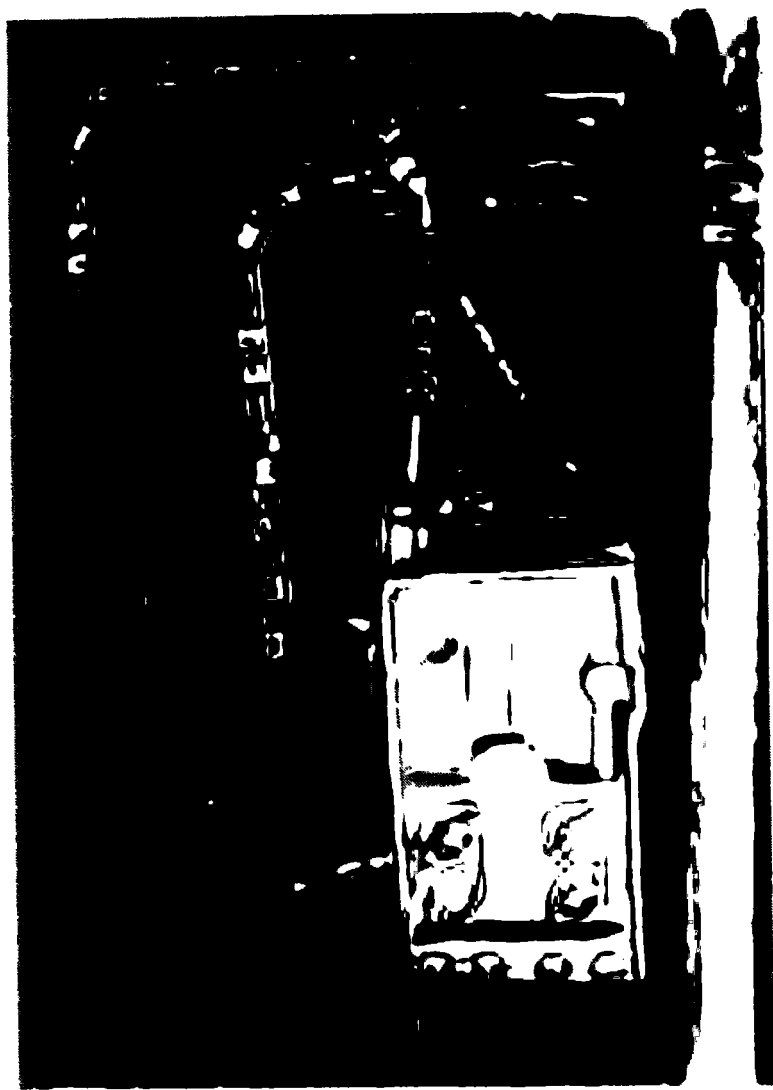
Fig 130. КУ-57 (KU-57) cutter-loader

The rear guides of the loop jib receive a loading chain carrying both cutting picks and loading flights to break up the coal web as it comes out of the jib and to load it onto the face conveyor. A plough-share and mouldboard also assist in the loading.

The cutter-loader is driven by an MAO-60/400 electric motor with a 60-minute rating of 80 kW.

M-81 supports are hydraulically-powered units in the form of sections having roof shields which are joined to each other by tongue-and-groove interconnecting plates. To close off the working face space from subsiding debris on the goaf side, telescopic rear shields are hinged to the roof shields.

Each support section is 800 mm wide and is backed by two hydraulic jack props hinged to the roof shield.

At their footplates, with which the jack props bear on the floor, the props rest on a spherical surface and can be inclined from the vertical when the support section must be moved over. The sections are moved over by the hydraulic rams mounted under the shields between adjoining sections. On the face side, the roof area is supported by cap pieces with hinged shield plates. Each support is individually controlled. Pressure oil for control of all the support sections is supplied from a common pump unit stationed at the gate end.

In addition to the mining systems with powered roof support systems described above, a series of other systems have been designed and tested in a variety of mining and geological conditions, including mining systems for pitching seams of different thickness, for seams of intermediate inclination, etc.

5-27. Coal Mining Units or Systems for Manless Extraction of Coal

The advent of mechanized roof support systems has paved the way for units or systems which fully mechanize all work at the face and are controlled remotely from a control panel box stationed in the gate road. These remotely operated systems may be called "manless" because they extract the coal in the absence of operators at the face.

The very first trials of the A-2 and A-3 remote-operated systems made it plain that manless extraction of coal was quite a practical proposition.

The A-2 system mechanizes all work at faces up to 60 metres long in seams of coal of any hardness 0.85 to 1.35 m thick with roofs of medium stability.

The system is made up of a toothed coal plough (or K-64 cutter-loader), a power-moved conveyor and mechanized hydraulic supports.

The scraper-chain conveyor, designed to be advanced as one unit, serves as the base for the entire system in which the coal plough, con-

veyor and roof control supports are interconnected into a single system.

Fig. 131 shows the A-2 system using a plough *1* which travels along the conveyor *2* with its body riding a guide tube *5* and its bracket riding the back sideboard of the conveyor. The coal plough is moved along the coal face at a speed of 0.73 m/sec by a chain whose return strand runs in the guide tube of the plough. The pull strand hauls the plough up from the gate end to slice a web from 40 to 100 mm thick
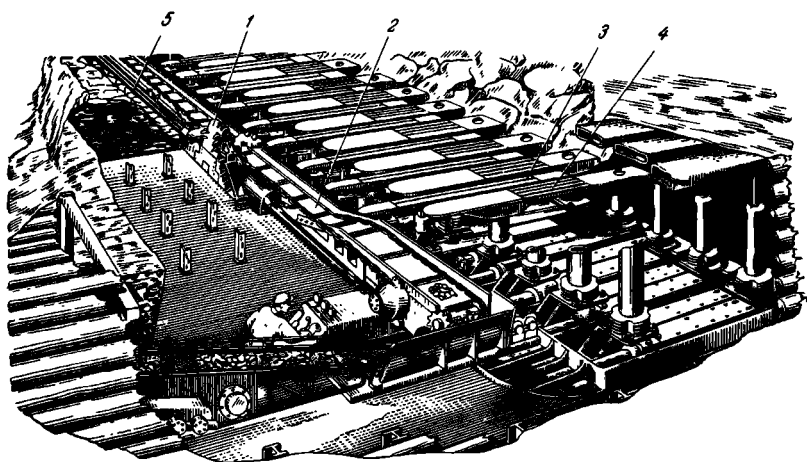
Fig. 131. A-2 remote-operated system

and load the coal on the conveyor. On its trip back to the gate end the plough clears the face and pushes the remaining coal onto the conveyor.

The pilot bars over which the tip of the piston rod of the hydraulic cylinder slides control the extensible part of the plough, thereby automatically adjusting the plough for height in accordance with the thickness of the seam being worked.

The A-2 system can operate with K-64 cutter-loaders having a drum type cutting head—a combination which will work seams of hard and tough coal. In this case, instead of the single plough unit, three narrow-web cutter-loaders will work the face.

The roof control supports of the system consist of face-roof *3* and waste-edge *4* sections. All the face roof sections are moved over concurrently with the conveyor frame, while the waste-edge chocks are moved 0.8 m either singly or in groups. For this, the waste-edge chocks are connected to the conveyor frame by hydraulic advance rams.

Switch-over for the return run at the end of the face, as well as switch-in for the cut along the face, is accomplished by the plough itself.

The design output capacity of the A-2 system is 44 to 65 tons per hour, with extraction of the coal taking place only during half the work cycle.

Four electric motors are used in the system and have a total power capacity of 161.5 kW. The weight of the system per metre of coal face is 2.68 tons. The total weight of the system is 73.9 tons for a face 25 metres long.

The A-3 system has been designed to remotely mine coal of medium hardness in seams from 1.5 to 2.1 metres thick with an easily collapsible roof (such as encountered in the Moscow coal fields). It works faces 50 metres long.

In the A-3 system the coal is cut from the face by a long-link endless chain fitted with flights and pick carriages carrying oversize picks. The picks shear the coal from the face to a depth of 80 mm and the flights transport it away from the face. Both of the chain strands run in a vertical plane at a speed of 1.04 m/sec. Chain drive is provided by two electric motors, each rated for 120 kW.

The roof supports in the A-3 system consist of line, intermediate and end sections. The bases of all the sections are hinged together to form a common base structure. On the face side, all the bases serve as the apron over which the coal is conveyed from the face. The weight of the system per metre of face is 3.9 tons.

In the Chertinskaya-Yuzhnaya pit (the Kuzbass coal fields) an A-3 system produced 6473 tons of coal per month from a face 17 metres long, the advance of the face for the same period being 147 metres. The maximum daily output reached 724 tons at a face-man output of 90.5 tons and a district-man output of 60.3 tons.

Apart from the A-2 and A-3 systems, efforts are being made to develop other methods of manless coal extraction, in particular, by hydraulic mining and also by coal ploughs and saws for use in pitching and other seams.

G. HEADING AND FACE-ENTRY MACHINES

Heading machines are used to drive development and haulage gate roads, while face-entry machines open up coal faces.

As to their field of application, heading machines divide into coal-heading, coal-and-stone heading and stone heading machines.

Existing heading machines can only operate in stone of not greater than medium hardness (up to a hardness of 4 on the Protodyakonov scale). Drivage in rock of greater hardness is carried out by getting the rock down by drilling and shot firing and then loading it with rock loading machines.

5-28. Coal Heading Machines

The ПК-2-1 or ПК-2м (PK) coal heading machine cuts full sections of gate roads in coal and consists, essentially, of a cutting head, gathering head, loader, and propulsion unit.

It is propelled on crawler tracks *1* (Fig. 132). When the machine is advanced to the face, its cutting chains, running in vertical jibs *2* and *3*, cut into the coal to a depth of 5 to 30 mm and rip it down as they travel from top downward.

While the chains so operate, the cutting head turning mechanism sweeps the jibs from one extreme side position to the other. The coal is thereby extracted across the full section of the heading. Each time the cutting head approaches its extreme right-hand or left-hand position at the face, the automatic feed control starts the crawler tracks (for a short interval of time) to feed the chains into the face as much as is necessary for the next cut (5-30 mm).

Coal ripped from the face by the picks falls on the gathering apron *4* from which it is taken by the scraper-chain conveyor unit mounted within the turning part of the cutting head assembly and is passed to a belt conveyor *5*. The latter can load the coal out into mine cars run under its jib or deliver it to a gate road conveyor.

The trapezoidal shape of the roadway (narrower at the roof and wider at the floor) is obtained by giving the cutting jibs a certain tilt during the side-to-side motion.

The machine is equipped with four electric motors: the main 32-kW motor drives the cutting chains, the mechanism which swings the cutting head across the face, and the scraper-chain conveyor unit; two 2.7-kW motors drive the crawler tracks; lastly, a 2.7-kW motor drives the loading belt conveyor.

This machine incorporates a water sprayer which aids considerably in the suppression of coal dust during operation of the machine.

This machine can drive roadways 5.1 to 8 square metres in cross section in soft and medium-hard coal beds at least 2.1 m thick and free from rock partings. In the U.S.S.R. this machine finds application in the Moscow coal fields.

The ПКС-3 (PKS-3) heading machine (Fig. 133) is a cutter-loader used to drive trapezoidal roadways from 5.12 to 7.4 square metres in cross section, primarily, for slice, bench and parallel headings in pitching seams of medium and large thickness.

The cutting head *1* of this cutter-loader has the form of a frame carrying the rotating boring arms arranged one above the other. Depending on the size of the heading to be driven, the boring arms may have an effective diameter of from 700 to 1100 mm. The arms are set at 10 degrees from the vertical in order to drive headings of trapezoidal cross section. The top and bottom boring arms are rotated counter-
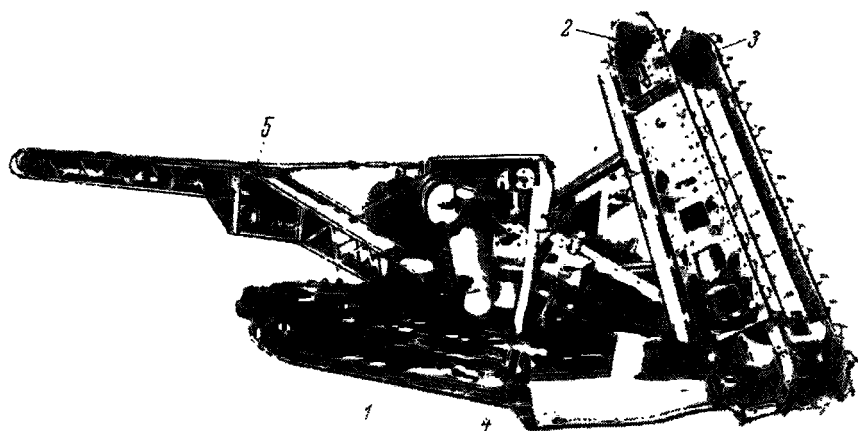
Fig. 132. ПК-2м (PK-2m) heading machine
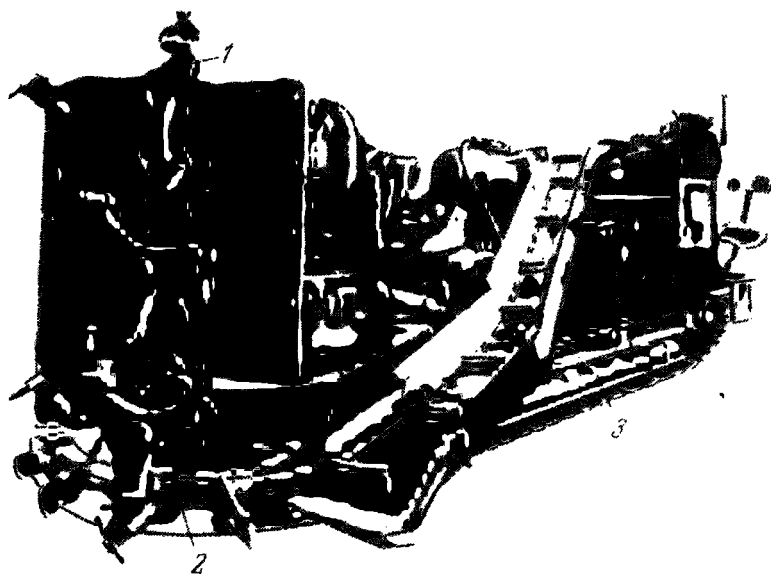
Fig. 133, ПКС-3 (PKS-3) heading machine

clockwise, the middle arm, clockwise. In operation, the boring arms rotate and are simultaneously swung from side to side across the face by hydraulic rams (the cross feed drive). As the coal falls, the loop scraper-chain conveyor 2 on the superstructure of the machine loads it out either into mine cars or onto a gate conveyor. The loop scraper-chain conveyor has a double-hinge link chain fitted with cantilever flights and driven by its own motor.
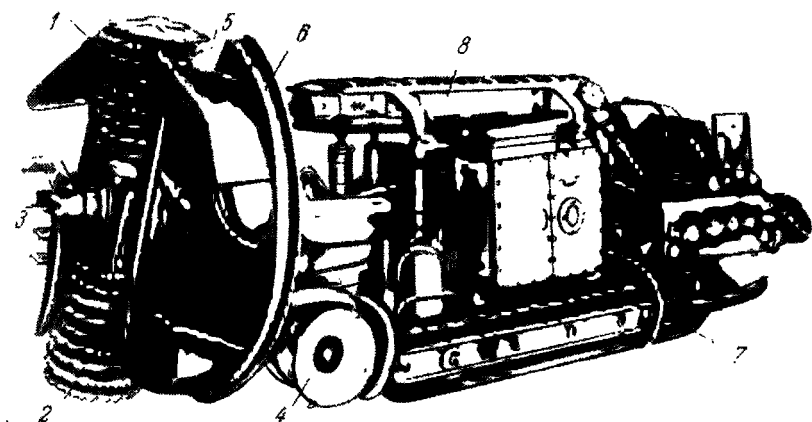
Fig. 134. ПКГ-3 (PKG-3) heading machine

The ПКС-3 (PKS-3) cutter-loader travels on crawler tracks 3. The earlier ПКС-2 (PKS-2) machine used a walking type drive powered by hydraulic rams.

The ПКГ-3 (PKG-3) heading machine (Fig. 134) is used to drive arched headings with a cross sectional area of 4.32 square metres in coal or rock of medium hardness when the seams are at least 2.5 m thick.

The cutting head of this machine is of multipick milling arm design with planetary rotation of the cutting tools. On two shafts 1 and 2 are mounted sets of opposing milling discs fitted with cutting picks. At the centre where the shafts converge a small boring arm 3 is also incorporated.

The milling arm shafts and the boring arm in the centre are rotated about the longitudinal axis of the machine while the shafts simultaneously revolve in opposite directions about their own axes. This motion makes the cutting picks on the discs cut annular grooves in the face and leave solid rings of coal in the face which are sheared out by the bits fitted in the sides of discs.

This head cuts circular headings which are given the arched shape by two benching millers *4* built as horizontal spiral drums fitted with picks.

The coal removed by the cutting head is gathered from the face by four buckets *5* attached to a driving trunnion. The upper part of the shield shutting off the space behind the cutting head has a hole and chute which transfer the coal onto the cutter-loader scraper conveyor. From the latter the coal drops onto a belt loader unit up to 20 metres long.

The ПКГ-3 (PKG-3) cutter-loader is mounted on crawler tracks *7* which travel along the two paths cut by the benching millers.

To attain good stability and ensure a high thrust pressure against the face, the machine incorporates a top thrust crawler track *8* by which it jacks itself between the floor and roof with a set of hydraulic rams. Drive for the upper crawler track comes from the cardan shaft taking power off at the main motor gear box.

The ПКГ-4 (PKG-4) heading machine is intended for driving headings 2.8 square metres in cross section in pits using hydraulic mining methods (to open up district airways, gate roads, do cross holing, etc.).

It has a triple-arm rotating cutting head. The coal, as it is broken down by the cutting head, is flushed away by water supplied to the machine by pipelines.

5-29. Heading Machines for Mixed Workings

The ПК-3 (PK-3), ПК-4 (PK-4) and (ПК-5) (PK-5) heading machines can extract rock of up to medium hardness under conditions, for example, such as are met with in the pits of the Moscow, Chelyabinsk and Lvov-Volynsk coal fields.

The cutting unit of the ПК-3 (PK-3) machine (Fig. 135) is a conical head *1* fitted with cutting picks on the sides and a boring nose at the top. This head has its picks fixed in a helical line round the conical sides and is secured to the end of a revolving arm which, together with the electric drive motor and the reduction gear, can sweep the heading both vertically and horizontally under the control of hydraulic rams. The head can therefore be set in any direction.

To work the face, the machine is set on the centre line of the roadway, the head is raised so that it is either in the right-hand or left-hand corner of the face, and the crawler tracks *3* are started to feed the rotating cutting head into the face to almost its full length. Then the head is made to sweep gradually across the entire face by the hydraulic rams. The coal broken down by the cutting head is gathered by the flights of the loop conveyor *2* for delivery through a

discharge throat onto a belt loader which drops the coal into mine
cars or onto a transport belt.

In the Chertinskaya-Yuzhnaya pit (the Kuzbass coal fields) a ПК-3
{PK-3) cutter-loader drove an entry 52 metres long in one day.

Fig. 135. ПК-3 (PK-3) heading machine

The ПК-4 (PK-4) machine has a longer arm. It also differs in that
the flight loop loader has been replaced by a jib with a single-hinged
scraper chain. This jib is laid on the inclined surface of the loader
feeder and is rigidly attached to the swinging platform of the machine.

The ПК-5 (PK-5) cutter-loader also has a longer cutting-head arm,
a wider angle of sweep, and electric motors of greater power rating.

Other modifications of these heading machines have been devel-
oped, such as the ПК-3В (PK-3V) with an extensible cutting head
arm; the ПК-6 (PK-6) for drivage of headings of small cross-sectional
area (4.32 sq m), and the ПК-7 (PK-7) for driving headings of 5.5
sq m cross section.

The ШБМ-2 (ShBM-2) heading machine drives arched tunnels for
single-track haulage roadways in hard coals, and in rock and mixed
seams of less than medium hardness.

When operated at a mixed face, the ШБМ-2 (ShBM-2) machine
breaks down the coal and the rock unselectively and therefore does

not provide the means for extracting the coal and the parting rock separately as do the ПК-3 (PK-3), ПК-4 (PK-4) and the ПК-5 (PK-5) machines.

The cutting unit of the machine (Fig. 136) is a triple-arm boring head *1* with an effective diameter of 3 metres. It is fitted with a set of picks which cut annular grooves 50 mm wide and 300 mm deep in the face and leave solid rings in the face, broken down by the wedge-shaped (conical) shearing picks attached on the arms behind the cutting picks.

As the coal or rock is cut and broken down, it falls into the space between the face and the guard shield *2*. Here a scoop *3* gathers the coal or rock when it is in the bottom position and discharges it through the window in the shield when it is in its upper position. The coal or rock falls on the belt loader for delivery directly into mine cars or onto another self-propelled loader.

The cutting head makes a circular opening which is given an arched shape by two benching cutters *4* fitted at the forward-end supports of the machine to form the side walls and also dint the floor for placement of the steel support arches.

The machine "walks" with the aid of four hydraulic rams, forward and rear supports, and a thrust girder *5*.

Two of the rams are of the jacking type and are built into the thrust girder to set it tight between the walls of the tunnel. The two other advance rams *6* have their cylinder ends attached to the main gear casing *7*, while the ends of their connecting rods are attached the thrust girder.

The advance rams move the machine forward, while the thrust girder is held immovable between the sides of the tunnel. After this, the pressure in the jacking rams is released and the thrust girder is freed from contact with the walls to permit the advance rams to push it forward for a new cycle of operation.

The cutting head and the two benching cutters are driven by the main 75-kW motor *8* through the main gear box *7*. Also mounted on the machine are the 4-kW motor of the belt loading conveyor, the 11-kW motor of the ditch digger *9*, the 4-kW motor of the dust exhauster *10*, and the 3.5-kW motor of the water spraying equipment.

5-30. Face-entry Machines

For mechanizing the drivage of rise entries, cross holes, face roads and other heading work, the Giprouglemash (the coal-mining machine design establishment in the U.S.S.R.) has developed the KH-1 (KN-1) and the KH-2 (KN-2) face-entry machines for seams 0.6 to 1.3 metres thick.
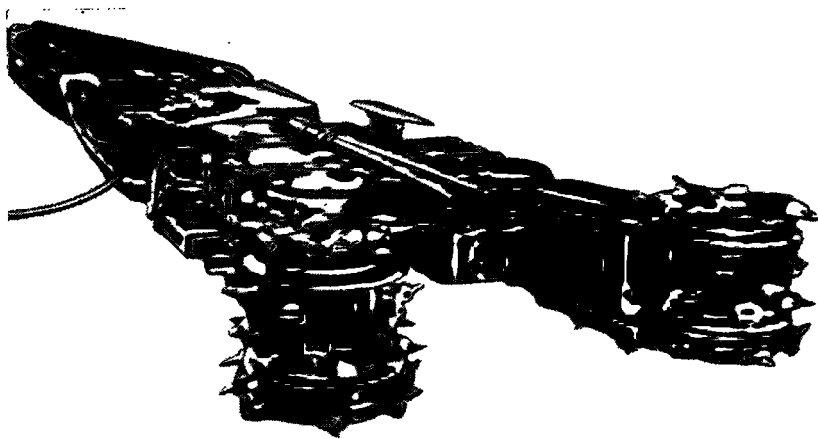
Fig. 137. KH-1 (KN-1) face-entry machine

The cutting end of the KH-1 (KN-1) machine has two jibs which operate at an angle to one another and are fitted with cutting chains. The latter revolve towards each other. Each jib is also fitted at the end with cutting discs carrying sets of picks (Fig. 137).

When the machine is in operation, the jibs are slewed from the sides of the heading towards the centre and back again by means of a hydraulic ram and rack mechanism.

As the coal is extracted, it is gathered by the loop scraper conveyor unit of the machine and is fed to a scraper installation or onto a conveyor for delivery to a loading point.

This machine uses an MA-191/10k electric motor for its drive.

The KH-2 (KN-2) machine (Fig. 138) uses the same haulage unit as the КМП-2 (KMP-2) coalcutter.

The cutting head of the machine consists of a top and a bottom jib in which a common double-hinge link cutting chain runs.

During operation of the machine the top jib sweeps the heading face from floor to roof and back again. The amplitude of sweep is adjustable between 0.7 and 1.2 metres, depending on the seam thickness. As this jib sweeps the face, its picks (on the upper chain strand) take down a web of coal.

The coal removed by the chain picks is then gathered by the lower strand of the cutting chain and passed to the loading side-mounted flight scraper conveyor arranged alongside the frame of the machine. From the loading conveyor, the coal is loaded out either by a scraper installation or another conveyor extended up to the machine as it advances the face.
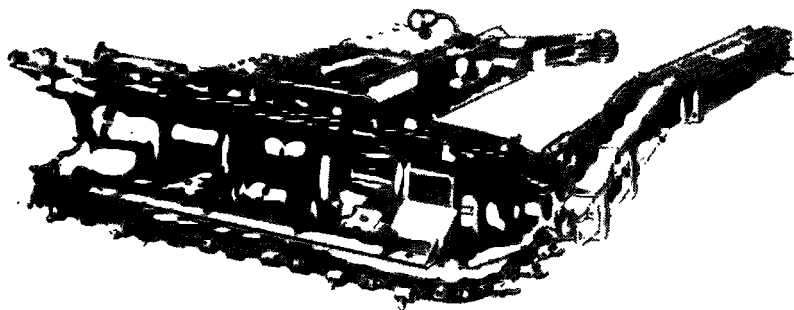
Fig. 138. KH-2 (KN-2) face-entry machine

Haulage is by rope.

When tested in a pit of the Krasnoarmeisk Division in the Donbass coal field, the KH-2 (KN-2) machine advanced a heading 8.5 metres in one shift. The design advance for this machine is 10 metres per shift.

5-31. Wide-work Heading Equipment

In the mining of thin and low-height seams, a great deal of refuse is produced in driving the headings. At some pits from 300 to 350 tons of refuse have to be loaded out to the surface for every 1000 tons of coal produced, an amount that places a heavy burden on the transportation facilities and winding equipment, and also creates definite handling and disposal difficulties at the surface.

If wide-work mining methods are used, a very considerable amount of refuse can be left underground. By these methods gob areas are maintained along the haulage gate roads for disposal of refuse produced during drivage of the headings.

To transport and stow the refuse in the wastes, it is common practice to make use of scraper installations and of hydraulic stowing equipment.

The ПЗУ-1 (PZU-1) scraper installation consists of a rock scoop or scraper and a СБЛ-4-2 (SBL-4-2) scraper winch. The unit is generally operated in conjunction with an ЭПМ-1 (EPM-1) or ППМ-5 (PPM-5) rock loading machine and a УПЛ-1c (UPL-1s) loader. In the operation of the above equipment the rock loading machine gathers the broken rock at the heading face to transfer it to the

loader, The latter drops the rock on the loading apron of the scraper unit for disposal to the wastes. The winch can then, by means of its pull ropes, run the loaded scraper into the wastes where it dumps the rock and then reverses the pull for return of the scraper. After the scraper has made two or three trips, it can be made to pack the rock more tight under the waste roof by butting the pack with its bridle bar.

For hydraulic stowing, the rock is transported from the face to a rock crusher, from which a water pumping unit or elevator hydraulically transports it into the wastes.

Coal in wide-work operations is extracted by the КШХВ (KShKhV) cutter-loader. It uses the frame of the face scraper-chain conveyor set up along the face for its track. The cutting unit of this machine comprises two spiral discs armed with cutting picks. These spiral discs both cut and load the coal on the conveyor. They are mounted at both ends of the machine, with the electric motor set in the middle. Being double-ended, this machine is shuttle-operated. Its spiral discs have a diameter of 650 mm across the picks and the depth of the web they can cut ranges from 0.5 to 1 metre. To cope with seams of different thickness, the height of the discs can be set anywhere between 0.6 and 1.3 metres.

At the end of a cut the conveyor and cutter-loader are pushed forward by hydraulic rams on a new line at the face. The cutter-loader discs are started at the same time and cut into the face. Thus no stables have to be excavated for this cutter-loader beforehand.

H. LOADING MACHINES

The major time and labour consuming operation in heading work is the loading out of the broken coal and refuse. When loading machines are used for this work, the rates of advance increase three to four times over those attainable by hand loading.

Loading machines are classified according to:

(1) Application; coal loading and rock loading machines. This classification is somewhat arbitrary since rock loading machines can be used to load coal, while coal loading machines may sometimes be used in mixed headings where they have to handle both the coal and the rock. A separate sub-group covers machines designed for work in inclined headings;

(2) Principle of operation of the loading arrangement; machines are grouped as continuously-operating [the УП-3 (UP-3), ГНЛ-30 (GNL-30), ПМУ-2 (PMU-2), МПЛ-1 (MPL-1), МГЛ-1 (MGL-1), МГЛ-3 (MGL-3), and ГПС-70 (GPS-70)], and as intermittently-operating or bucket type [the ЭПМ-1 (EPM-1), ПМЛ-5 (PML-5), МПУ-1

(MPU-1), МПК-1 (MPK-1), and ППМ-5 (PPM-5)]. Machines which do the loading by means of buckets usually serve as rock loaders;
(3) Travel; either on crawler tracks or wheels;
(4) Drive; electric and pneumatic.

5-32. Coal Loading Machines

The УП-3 (UP-3) loading machine (Fig. 139) consists of a gathering unit *1*, crawler tracks *2*, a scraper-chain loading conveyor *3*, electrical equipment and a hydraulic system.

The machine has two electric motors; one of them is a КОФ 21-4 (KOF 21-4) motor of 15 kW rating which drives the gathering arms and scraper chain, and the other, an 11-kW КОФ 12-4 (KOF 12-4) motor which drives the crawler tracks and the hydraulic system pump.

Raising and lowering of the gathering nose pan and the conveyor boom, turning of the conveyor boom to the right and left, and shifting of the double friction clutches in the crawler track reduction gear is done by hydraulic rams.

The УП-3 (UP-3) loading machine is an improved modification of the C-153 (S-153) machine. It operates by gathering and feeding the coal to its scraper-chain loading conveyor with two mechanical arms. The gathering unit is so constructed that it can load both coal and rock. For dust suppression, the gathering head has also been fitted with water spraying devices. Since the ground pressure of the УП-3 (UP-3) machine crawler tracks is less than the pressure of the C-153 (S-153) machine tracks, the УП-3 (UP-3) machine can work on much weaker ground.

The ГНЛ-30м (GNL-30m) coal loading machine is mounted on crawler tracks and is a lightweight unit of 30 tons per hour loading capacity. It is built on the same principle as the C-153 (S-153) machine, but it is smaller in size and weight and can therefore work in headings of smaller cross-sectional area than the S-153 machine. Another difference in this machine (Fig. 140) is that it has a receiver scraper-chain conveyor *2* onto which the coal or rock is fed by gathering arms *1*. From the receiver conveyor, the loaded coal or rock is dropped on the loading belt conveyor *5* for discharge into mine cars or onto some other means of transport such as another conveyor. The loading belt conveyor can be swung 30 degrees to the right or left in a horizontal plane, and can also have its boom end raised or lowered through up to 16 degrees.

Four electric motors, each of 2.7 kW rating, are mounted on the ГНЛ-30м (GNL-30m) machine; the first to drive the gathering arms and the scraper-chain conveyor, the second to drive the belt conveyor and the hydraulic system oil pump, and the last two to drive the crawler tracks *3*.
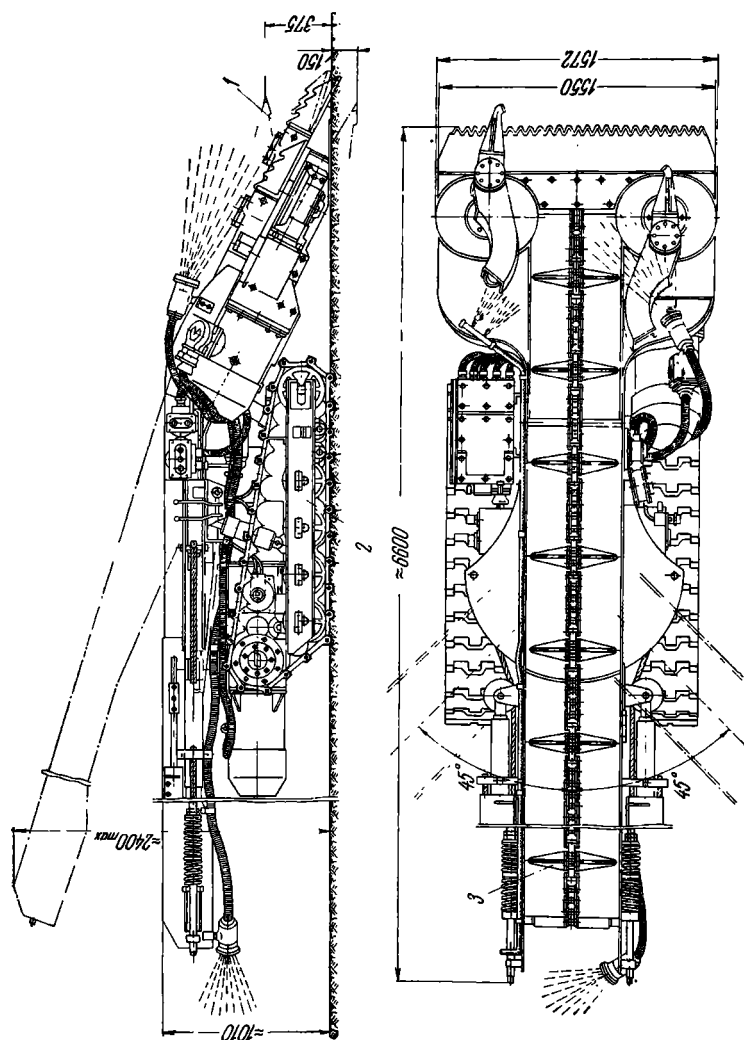
Fig. 139. УП-3 (UP-3) coal loading machine
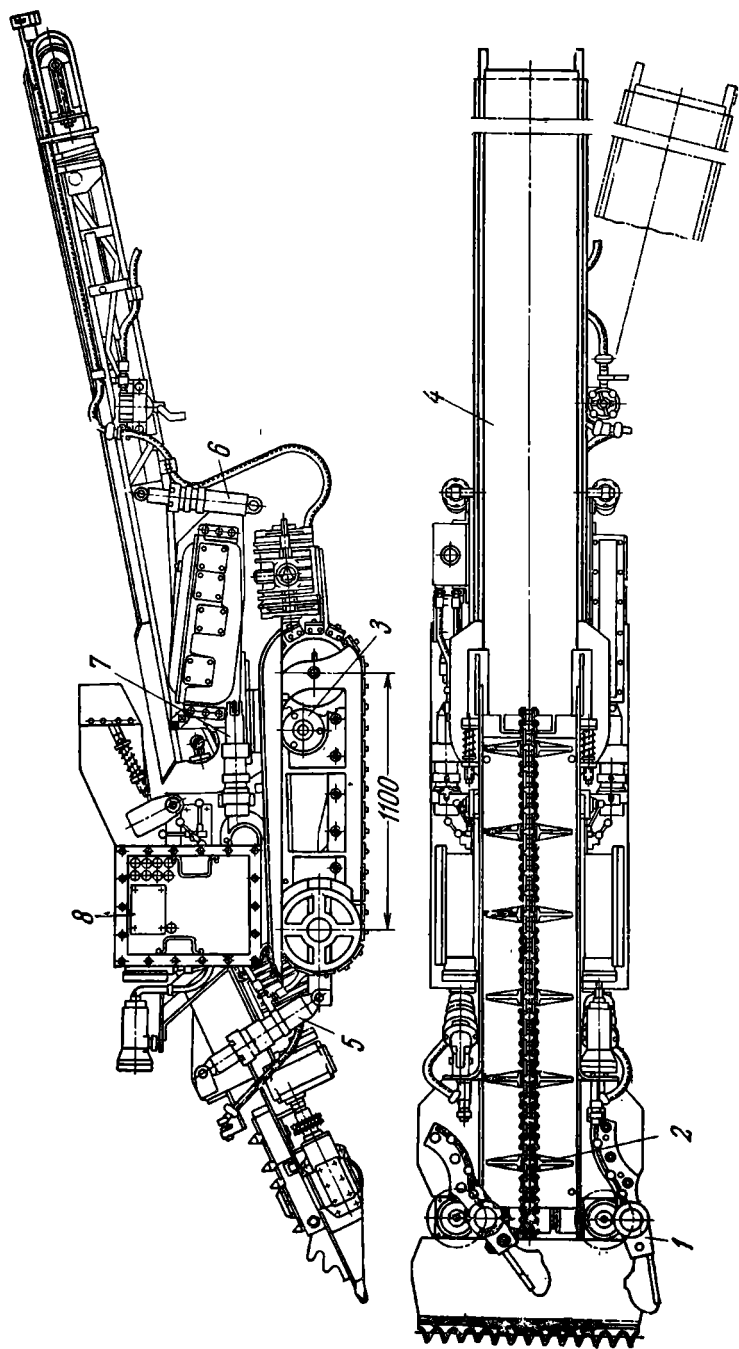
Fig. 140. ГНЛ-30м (GNL-30m) coal loading machine

All the motions, such as raising and lowering of the gathering end, and raising, lowering and swing-over of the belt conveyor, are performed by hydraulic rams *5, 6* and *7*. For local lighting at the heading face, the machine is furnished with a headlight. Control of the motors is accomplished through a magnetic starter box *8*.

The reduction gear box of the gathering arm drive, the gathering arm parts, and the conveyor drive require lubrication with grade L machine oil at intervals of one week, but the crawler tracks and the scraper chain must be oiled every day.

Grease must be added once a week to the track and support roller bearings, the bearings of the conveyor-chain tensioning device shaft, and the track-sprocket shaft bearings. At monthly intervals grease must be added to the bearings of the tension end drum, gathering apron feet, the conveyor driving and tail-end drums, and the conveyor idlers.

The hydraulic rams must be constantly maintained lubricated when they are in operation.

The **ГПС-70 (GPS-70) coal loader** is a crawler-mounted self-propelled unit of 70 tons per hour loading capacity designed to meet the conditions of the Kuznetsk and Karaganda coal fields and eventually supersede the ГНЛ-30м (GNL-30m) loading machine. The ГПС-70 (GPS-70) machine has a greater loading capacity and a gathering unit suitable for handling lump coal and rock up to 700 mm across with no possibility of the hard material jamming between the gathering arms and the scraper flights. Drive for the truck and belt loading conveyor is provided by a 2.7-kW ВАД-27 (VAD-27) electric motor. A КОМ-22-4 electric motor of 2.8 kW rating is used to drive the gathering unit.

МПЛ-1 (MPL-1), МГЛ-1 (MGL-1) and МГЛ-3 (MGL-3) small-size loading machines do not differ in principle from the coal loading machines described above. They incorporate two gathering arms and are mounted on crawler tracks.

In the МПЛ-1 (MPL-1) a receiver scraper-chain conveyor is used between the gathering unit and the loading belt conveyor. Both the gathering unit and the belt conveyor may be lowered or raised by a system of hydraulic rams.

The МГЛ-1 (MGL-1) machine mounts a belt conveyor and a loader unit arranged so that its receiving end is hinged to the conveyor jib and can be turned through 45 degrees in a horizontal plane either to the right or left of the longitudinal axis of the machine. At the discharge end the loader unit has a supporting hopper carriage seated on the longitudinal guide rails of the face conveyor. This carriage has enough free travel to allow the machine to advance 1.5 metres forward.

The МГЛ-3 (MGL-3) machine has a receiver and a loading belt conveyor. Since the loading conveyor is hinged on the truck frame of the machine, it can be swung through a horizontal angle of 100 degrees to the right or left.

5-33. Rock Loading Machines

The ЭПМ-1 (EPM-1) rock shovel loader is an electrically driven machine designed to load coal and rock in the drivage of horizontal workings. It (Fig. 141) consists of a wheel-mounted undercarriage driven by a separate motor, a slewing body *1* turnable on a spherical support set on the frame, and a digging-rocking unit *2* carrying a shovel *3*. A second motor provides the power for raising and throwover of the shovel. The control panels are mounted on the swivel body.

The successful operation of the shovel loader depends on the lump size it has to handle. This, in turn, involves the proper planning of shot-hole drilling and firing. In order to keep the lump size of the broken-down material less than 300 mm, the shot hole drilling and firing must be done to the pattern approved by the mine administration for the given face. If the lumps are too big, the output of the loader drops sharply.

Fig. 141. ЭПМ-1 (EPM-1) rock loading machine

The loader operates as follows. The machine is run within 1.5 metres of the heap of debris at the face, the shovel is lowered and is run into the heap. To fill it with more material, the shovel-lifting motor is jogged several times. As soon as the bucket becomes filled, the machine is backed away, the shovel is swung upward and over to discharge its load into a car or tub behind the loader. The car is coupled to and travels together with the loader until it is filled.

Traction power is transmitted from the drive motor to both wheel pairs through a reduction gear. A two-stage reduction gear transmits power from a separate motor to drive the chain drum by which the

Fig. 142. Right-hand control panel of ЭПМ-1 (EPM-1) rock loading machine

shovel is elevated. The chain is attached to the drum at one end and to the shovel at the other. In elevating the shovel, the chain winds spirally on the drum.

Machine Controls. Right-hand and left-hand control panel boxes are provided on the swivel body. They are flameproof, house all the control apparatus, and have the push buttons brought outside.

The right-hand box (Fig. 142) comprises two КТД-2АЗ (KTD-2AZ) contactors *1*, starting resistors *2* and push buttons *3, 4, 5, 6, 7* and *8*.

The left-hand box (Fig. 143) comprises two КТД-2АЗ (KTD-2AZ) contactors *1*, transformer *2* for supply of the lighting circuit, and push button stations *3, 4, 5, 6, 7* and *8*.

The push buttons on the right-hand box are connected in parallel with the push buttons on the left-hand box for control from either side of the machine. Push buttons *3* on each box control the forward travel of the machine, and push buttons *4* the rearward travel. Push buttons *5* energize the shovel elevating motor through a resistor to reduce the shocks incident to frequent starting; push buttons *6* start the shovel-elevating motor by direct connection to the line. Push buttons *8* are mounted inside the boxes (they de-energize the gate-end box magnetic starter when the control box cover is removed). The forward and rearward travel control contactors are so interlocked that when one of them is closed, it first breaks the operating coil circuit of the second contactor by opening its normally-closed interlock. This prevents the two contactors from closing simultaneously. A short circuit would otherwise result if both push buttons were pushed simultaneously by mistake.

Machine Lubrication. The reduction gearing of the undercarriage and shovel-elevating mechanism is splash lubricated, Grade 38 cylinder oil being added as needed to maintain the oil bath at the right level. Every 3 to 6 months the oil in the gear cases must be replaced.

The upper chain guide roller and the automatic turnover mechanism sleeve bearings are lubricated with УС-2 (US-2) grease once a shift by means of a grease gun.

Oiling of the hinge joints in the automatic turnover mechanism is done every shift, while the chain is oiled throughout its length once a day with Grade 11 cylinder oil.

All the antifriction bearings must have УС-2 (US-2) grease added at monthly intervals.

The ЭПМ-2 (EPM-2) rock shovel loader is a new prototype developed by the Toretsk Engineering Works. Its characteristics are the same as those of the ЭПМ-1 (EPM-1) machine, but it has a travelling remote control panel and fluorescent lighting fittings.

The ПМЛ-5 (PML-5), МПУ-1 (MPU-1) and МПК-1 (MPK-1) Rock Shovel Loaders. Pneumatically powered, the ПМЛ-5 (PML-5) is intended to handle shot-fired broken rock and coal at the faces of

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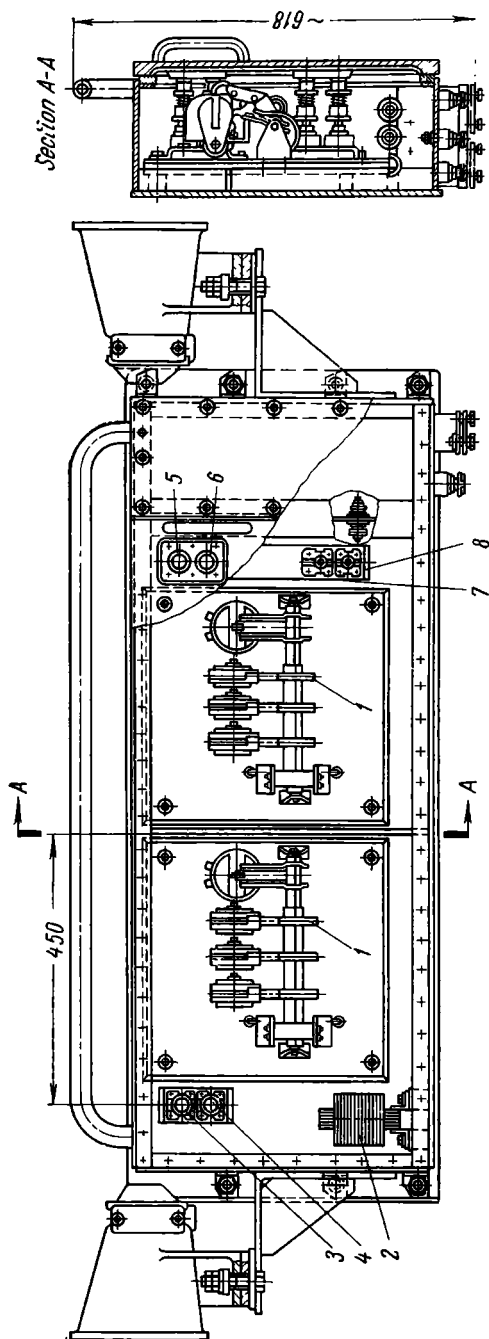
Fig. 143. Left-hand control panel of ЭПМ-1 (EPM-1) rock loading machine

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horizontal headings where power is derived from compressed air.

In design and principle of operation, the ПМЛ-5 (PML-5) is similar to the ЭПМ-1 (EPM-1). Instead of electric motors, it is fitted with two compressed-air motors, one to drive the running gear, the other to elevate and lower the shovel. This machine has found its widest application in the Donbass coal mines with thin pitching seams.

Another prototype developed by the Toretsk Works is the МПУ-1 (MPU-1) which can use either compressed-air or electric drive. With a loading capacity from 1.5 to 2 times that of the older models, the МПУ-1 (MPU-1) will with time replace the ЭПМ-1 (EPM-1) and ПМЛ-5 (PML-5) machines. Moreover, the new machine will require less headroom for shovel turnover and therefore be able to work in low headings. The loading width will also be greater.

The МПК-1 (MPK-1) is a machine of relatively small dimensions for work at headings more than 1.8 metres high.

It differs from the ЭПМ-1 (EPM-1), ПМЛ-5 (PML-5) and МПУ-1 (MPU-1) units in that it incorporates an apron-scraper loading conveyor bracket-mounted on the truck frame of the machine. It unloads its shovel on this conveyor and the latter discharges into mine cars. Three compressed-air rotary-type motors drive the shovel, running gear, and loading conveyor.

The ППМ-4 (PPM-4) and ППМ-5 (PPM-5) rock loaders are employed to load shot-fired rock and coal into mine cars or onto conveyors in horizontal headings of large cross-sectional area. In headings up to 4 metres wide these machines will load across the face from one track, and in headings wider than 4 metres, from two tracks in turn.

These machines have replaced the УМП-1 (UMP-1) units, produced up to 1952 and widely used in coal pits, mainly in double-track workings.

The machine operates as follows: with its shovel lowered and a mine car coupled behind it, the machine runs forward to crowd the shovel into the heap of debris, scoops up the material in the shovel and, with an upward and over motion, discharges the loaded shovel into the receiving hopper of the loading conveyor. As soon as the shovel is emptied, it is again dropped down, and the machine withdraws a little before it pushes into the heap again.

The main features of the ППМ-5 (PPM-5) loader may be seen in Fig 144. At the gathering end it has a shovel *1*, elevated and lowered by chains *2* wound on the elevating drum.

The welded underframe *3* is wheel-mounted. Conveyor *4* is individually driven by an electric motor *5* and comprises head- and tail-end drums, a set of idlers and a belt. Belt tension is maintained by adjusting the tail-end drum position with a telescopic screw arrangement.

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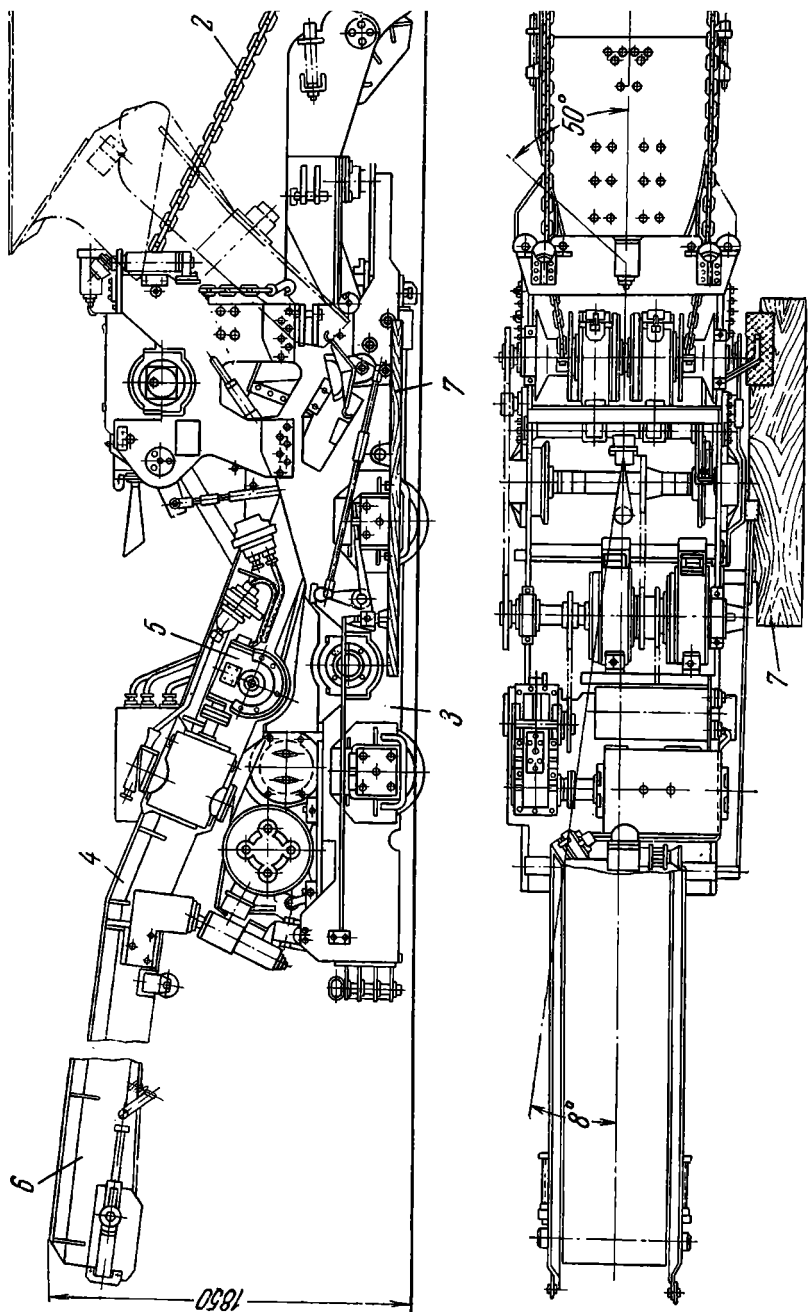
Fig. 144. ППМ-5 (PPM-5) rock loading machine

The conveyor can be swung 12 degrees to the right or left in a horizontal plane and its discharge end *6* can be raised 500 mm by means of a lifting arrangement.

The running gear and the shovel elevating mechanism are driven by the main motor via the reduction gear. The operator stands on a footboard *7* and runs the machine by manipulating the control levers arranged on the same side.

The ППМ-5 (PPM-5) uses an intermediate boom-chute to receive the rock from the shovel. This chute can hold from three to four shovel-loads of rock, a feature materially raising the output capacity of the machine. Three electric motors are mounted on the machine; one for conveyor drive, one for raising, lowering and swinging the shovel, and one for wheel drive.

5-34. Loading Machines for Inclined Workings

The **ПМУ-2 (PMU-2) machine** is used for loading rock and coal into mine cars or onto conveyors in inclines or headings with angles of inclination as great as 20 degrees where the heading cross-sectional area in the clear is not less than 4.5 sq m and the headroom is at least 1.9 m as measured from the top of the track rails.

A gathering head of the continuous-acting type is used in this machine, consisting of two gathering arms driven through crank mechanisms of the same kind as used in the C-153 (S-153), ГНЛ-30м (GNL-30m), ГПС-70 (GPS-70) and the УП-3 (UP-3) loaders. Two apron trough-type conveyors, one receiving, the other loading, are incorporated. Their conveying surfaces are formed by partly overlapping plate members which therefore do not allow the material to drop out. The truck in the ПМУ-2 (PMU-2) machine is a track-wheeled self-propelled underframe fitted with a winch, and front and rear spring-type rail clamps. Travel towards the face (downhill) at inclinations greater than 12 degrees is accomplished by gravity, with the winch rope drum drive disengaged and the band brake somewhat released. For travel uphill, the winch drum drive is engaged to take up the rope on the drum and haul the machine. The rail clamps hold the machine fixed on the track rails whenever the rock and coal are being loaded in order to take the sideward reaction of the gathering arms.

The welded frame carrying the gathering head and the receiving conveyor is mounted on the truck underframe at an angle of 23 degrees.

Raising, lowering and turning of the receiving conveyor and gathering arms through an angle of 25 degrees to one or the other side of the centre line is carried out by hydraulic cylinder rams. From a single position of the head, the machine gathers coal or rock over a
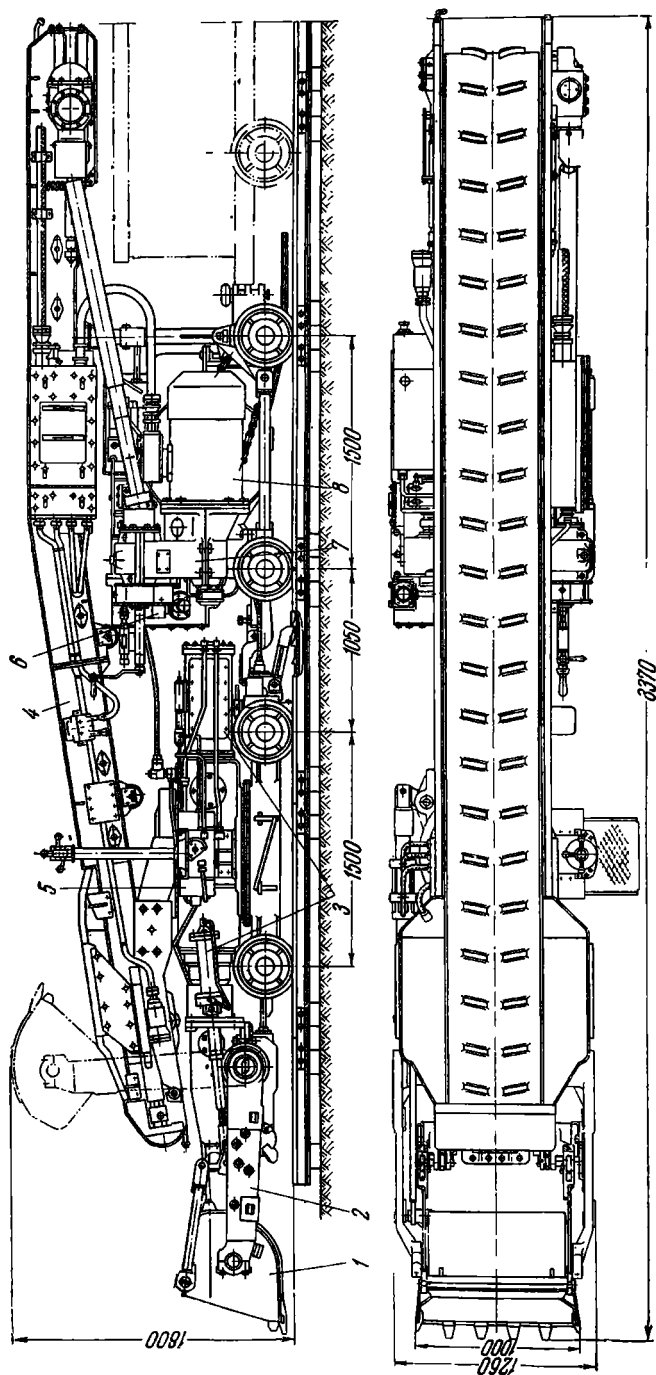
Fig. 145. Prokhodchik rock loading machine for inclined headings:

1—shovel; *2*—shovel leverage; *3*—hydraulic cylinders; *4*—conveyor; *5*—distributor box; *6*—hydraulic system pumps; *7*—drive unit gear box; *8*—electric motor

width of 1.8 metres. To load out the material from across the full width of the heading, the machine must make two to three approaches, each time with the gathering head turned through the respective angle.

Three electric motors are used on the machine; one drives the gathering arms and the receiving conveyor, the second is for wheel drive, and the third drives the hydraulic system pump.

For versatility, the machine is fitted with two manipulators which can mount post-supported electric drills or air hammers. Spring-type rail clamps are provided to hold the machine in position if the haulage rope breaks.

The Prokhodchik (Fig. 145) is a loading machine intended for use in workings with inclinations up to 25 degrees, but it can be used in the drivage of horizontal entries as well.

The Prokhodchik is a shovel loader in which the shovel is operated by a leverage system actuated by hydraulic cylinder rams.

In loading, the machine is stationary and the shovel is pushed into the shot-fired rock by the hydraulic rams. The shovel then unloads automatically. Downhill and uphill haulage of the machine is provided by the winch mounted on the drive end of the machine which is connected to the gathering end by a pin-type car coupler. The drive end also mounts the belt-scraper loading conveyor with a hopper into which the shovel discharges the rock. Both units of the machine are mounted on wheel-mounted truck frames.

5-35. *Care of Loading Machines. Rules for Safety in Their Operation*

To avoid troubles, breakdowns and accidents, it is the primary duty of the machine operator and the district electrical fitters to be skilled in the lubrication and adjustment of the machine, keep constant watch on the operation of the machine, and follow the general rules for care and maintenance which may be summed up as follows:

(1) The machine should be regularly lubricated according to the schedule given in the manufacturers' instructions;

(2) Inspections and maintenance repairs should be done on the basis of an approved maintenance schedule;

(3) Before work is started, the machine must be checked for overall condition, especially for loose bolted joints, and correct operation of the control levers, clutches, motors and shovel mechanism;

(4) Motor enclosures must be cleaned of coal and stone dust and cuttings because deposits of dirt hinder the cooling of the motor and may lead to dangerous rises in temperature in the motor windings;

(5) Contacts must be watched, periodically cleaned of deposits and maintained smooth;

(6) Electrical connections must be watched and maintained in good condition and always reliably secured;

(7) Machines operated by compressed air must have their air mains and individual air distributing lines maintained air-tight at all the connections; air leakage means sharp loss in productivity of the machine. Air leakages may also be developed at worn slide plugs or valve discs in the air controls. In such cases the slide plugs have to be replaced and the valve discs reground to a close fit.

The essential rules for safety in the operation of loading machines are:

(1) The working in which the machine is operated must be maintained reliably supported, newly exposed roofs must not be allowed to become too great in area.

(2) Men not responsible for operation of the machine should never be allowed to stand near, or the more so, in front of it.

(3) It is never allowable to couple or uncouple a mine car at the machine when it is loading out coal or rock.

(4) No repairs must be attempted on the machine with the shovel raised, if it is not propped, or the electrical equipment is not disconnected from the supply.

(5) Earthing circuits must be checked for continuity and reliable connections with utmost attention.

I. MINE DISTRICT PUMPING UNITS

Coal mines, in addition to their main drainage pump installations, generally have supplementary installations for pumping the water from the district standages, inclines, dip workings, etc., into the main storage sumps.

Supplementary pumping of water is mainly done by centrifugal pumps and, less frequently, by reciprocating pumps.

5-36. Reciprocating Pumps

Principle of Operation

A reciprocating or piston pump (Fig. 146) consists essentially of a cylinder A, piston P, discharge valve V_2, and intake or suction valve V_1. Connected to the pump are a discharge or delivery pipe D and a suction pipe S.

When the piston P moves to and fro in the cylinder A, it makes the discharge and the suction valves open and close alternately.

As the piston travels from left to right, it produces a vacuum in the pump chamber on the left side; the water in the suction pipe,

under the force of the atmospheric pressure, then begins to rise, lifts the suction valve V_1, and fills the pump chamber and the free cylinder space.

On return to the right, the piston builds up a pressure which closes the suction valve V_1, opens the discharge valve V_2 and permits the piston to force the water out into the delivery pipe line. As the piston starts to move backward to the left on the next stroke, a new cycle of suction begins.

Most reciprocating pumps are today driven by electric motors through a crankshaft and connecting rods.
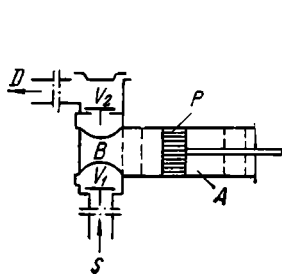
Fig. 146. Operating principle of single-acting reciprocating pump
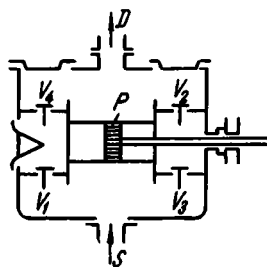
Fig. 147. Operating principle of double-acting reciprocating pump

A pump operating on the principle explained above is called single-acting. To get around the shortcomings of such a pump, such as a large cylinder and extremely irregular rate of delivery of water, the double-acting pump has been devised.

In a double-acting pump (Fig. 147) the piston P, as it moves from left to right, produces a vacuum in the left-hand side of the cylinder and sucks water from the sump through the suction pipe S and the left-hand suction valve V_1. At the same time, the piston builds up a pressure on the right-hand side, the water opens the discharge valve V_2 and discharges into the delivery pipe D.

As the piston moves back from right to left, the suction valve V_3 on the right and the discharge valve V_4 on the left open. Now, the water discharges through the latter valve into the delivery pipe D.

The valves are the most vital parts of a reciprocating pump; no pump can operate normally if its valves fail to function properly.

Each valve should fit closely to its seat so as to stop the water flow when closed, operate without undue noise, and freely pass water when opened. The mating surfaces of the valve plug and the valve seat must always be machined and ground to a smooth finish.

Air Vessels and Their Purpose

The action of a reciprocating pump is fundamentally intermittent and air vessels are fitted on the suction and discharge side to obtain a more uniform delivery of water and quieter operation of the pump by cushioning the intermittent forces. The larger an air vessel, the smoother the operation of the pump.

5-37. Centrifugal Pumps

Principle of Operation

In its essential form, a centrifugal pump comprises a casing A (Fig. 148) which houses a rotating impeller B carrying a set of vanes. The casing guides the fluid into the impeller through its suction inlet
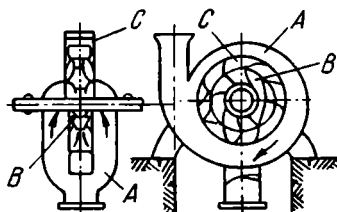
Fig. 148. Operating principle of centrifugal pump

and discharges it from the impeller through its spiral chamber (volute) to the delivery outlet. In Fig. 148 the pump casing has a set of stationary guide vanes to lower the hydraulic losses. By rotating the impeller at high speed after first priming the casing, the fluid is thrown from the centre to periphery of the impeller. This produces a vacuum on the inlet side of the pump and the fluid is caused to enter the pump and be discharged from it in a continuous stream.

In order to obtain high pressures, several impellers are mounted on the same shaft and pass the water consecutively from one to another.

The clearances between the impellers and the guide vanes should not be less than 0.5 mm, but not be very much greater.

As was stated earlier, when the pump is in operation its impeller produces a vacuum at the inlet, and the water rises in the suction pipe under the action of the atmospheric pressure. Since the atmospheric pressure can only sustain a water column 10.33 metres high in vacuum, the suction head (the vertical distance from water level in the sump to the highest point in the suction pipe) can theoretically never be higher than 10.33 m.

Reciprocating Pump Troubles, Their Causes and Remedies

Trouble	Cause	Remedy
Pump fails to work when started	Foot valve is poorly seated	Check and clean foot valve, grind mating faces to close fit, replace bad gasket
	Leaky flange joints on suction line	Check flange joints, pull their bolts tight, replace gaskets
	Excessive suction lift	Arrange suction line end so that vertical distance from pump axis to lowest point of water level does not exceed 6 to 7 m
	Seizure of valves	Check valves and grind them in
Flow stops during pump operation	Broken valve or piston; piston has lost its fit	Check valves; if a valve is broken replace it. If valves are in order, open rear cylinder cover and inspect piston and piston rings. Replace all broken parts. When piston rings are found to be too loose, replace them. Tighten gland or renew packing
Pump delivery is too low	Valves seat loosely, air leaks have developed in cylinder or gland	Eliminate all air leaks, add or renew packing in glands, clean valves, replace worn valves and seats
Pump operates with shocks and jerks	Air vessel on suction or discharge pipe is not filled with air	Fill vessel with air
	Slack or broken valve springs	Put in new springs
	Worn bearings	Check bearings, if necessary re-babbitt them
	Large clearance between piston rings and piston body	Put in new rings
Glands run hot	Glands pulled too tight, packing has lost its elasticity	Slightly back off gland nuts, renew packing
Bearings run hot	Poor lubrication or dirty oil	Check oil rings for proper running. Renew oil

The practical suction head is, however, only 6 to 7 m because air leaks into the suction pipe at the flange joints and gland boxes, and some of the vacuum is lost in the pipe. Other factors which reduce the suction head are the frictional losses at the bends and elbows in the suction line, strainer screens, foot valve, etc.

For any pump to deliver water to a standage, it must overcome the suction head, all the frictional resistances in the piping from the intake screen up to the standage basin and the lift head (the vertical distance from the pump axis to the axis of the delivery end of the discharge pipe).

In the mounting of pump installations, special care must be taken to make all the flange joints in the suction and discharge lines both air and liquid tight. These requirements likewise apply to all stuffing boxes and pump glands.

To make flange joints tight, it is necessary to place gaskets between them. They may be of rubber-asbestos composition material, rubber, rubberized fabrics, etc.

Start-up and Shutdown of a Pump

Before a centrifugal pump is started up, check to see that:
(1) Its bearings are properly lubricated;
(2) The glands are fully packed and tightened;
(3) The motor rotor and pump shaft are free to rotate when turned by hand;
(4) The fuses in the starter box are not blown.

When the check shows that the start-up may be made, the pump is primed with water from the discharge line (if it contains water) or through a funnel and the priming valve; the valve on the discharge line is checked to be sure that it is closed (in the case of a reciprocating pump the discharge valve must be open); the starter is switched on to start the pump; the pump is watched to see that it runs in the proper direction, and the discharge valve is opened for delivery of the water. The pump should not be allowed to run very long with the discharge valve closed.

When a centrifugal pump must be shut down, the discharge valve is first closed and then the motor is switched off.

The main, district and heading drainage installations in coal mines in the Soviet Union use КСМ (KSM) and АЯП (AYaP) multistage horizontal-shaft centrifugal pumps consisting of sectional impeller-casing units. The number of units employed in a pump is determined by the head against which the pump must operate. The casings and impellers of pumps used to handle neutral mine waters are of cast iron, those which have to handle corrosive and acid mine waters are cast of chromium-nickel steel.
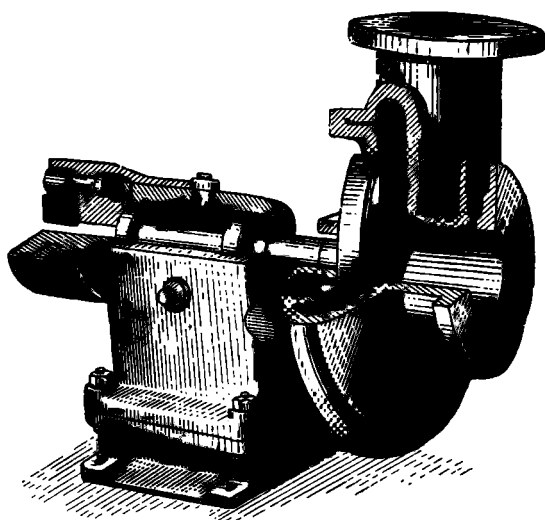
Fig. 149. General view of K-60 pump cut away to
show impeller

The above pumps are not altogether economical in operation be-
cause of their limited efficiency, heavy weight and large dimensions.
This has led to the development of Type MC (MS) multistage section-
al centrifugal pumps, the essential specifications of which are given
in Table 10.

Table 10

Pump characteristics	Pump type				
	MC-30	MC-50	MC-100	MC-150	MC-300
Rated delivery, m³/hr . . .	30	50	100	150	300
Water head per impeller, m	25	35	55	72	110
Motor speed, rpm	2950	2950	2950	2950	2950
Permissible suction head, m	8.6	8.37	6.8	5	—
Efficiency	0.63	0.66	0.7	0.72	0.75

As in the above pumps, the pump head, motor rating and weight
of a pump depend upon the number of impellers used in it.

A pump available for district transfer drainage is the K-60 overhung
centrifugal low-head pump. It is designed with an open type impeller
mounted overhung on the pump shaft (Fig. 149). The advantages of
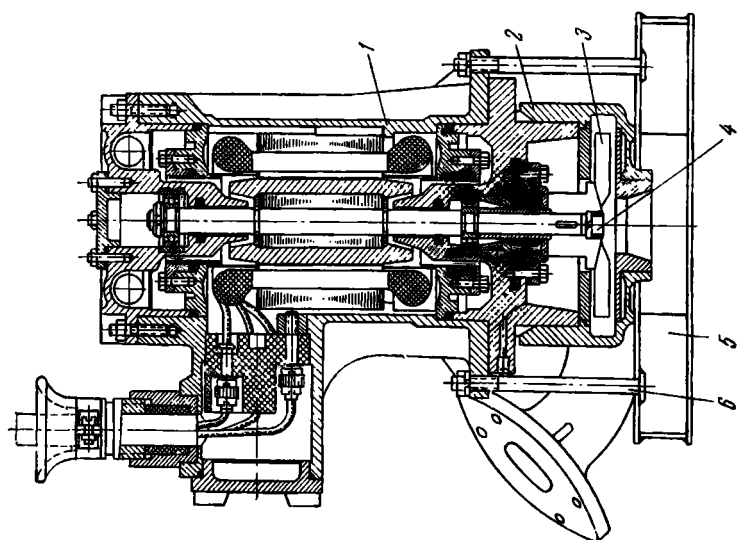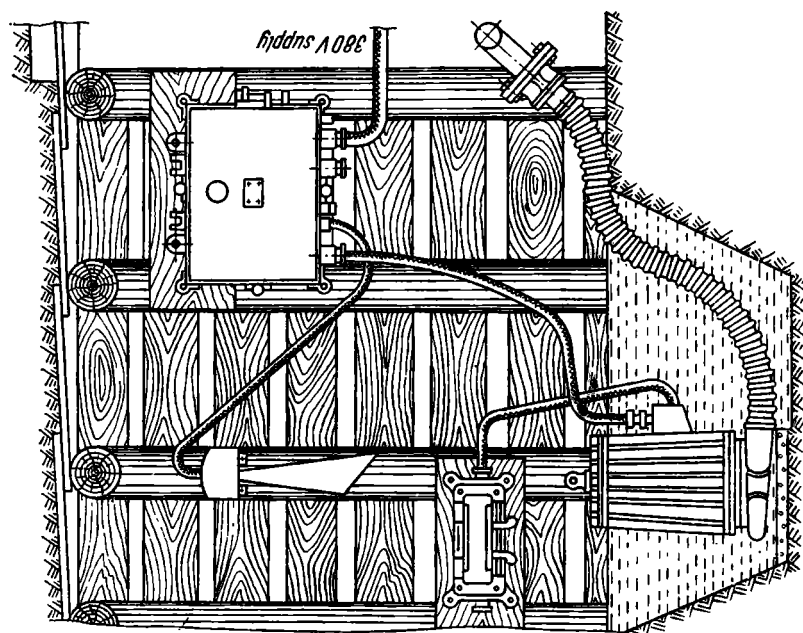
such an impeller become quickly apparent when dirty mine waters have to be handled. This pump must be run clockwise, as viewed from the drive end.

The pump shaft rotates in two ball bearings running in an oil bath provided in the pump pedestal. Each pump is furnished complete with an electric motor, a flexible coupling, and, on request, with a bedplate. This type of pump is the best for inbye transfer pumping.

Specifications of the K-60 pump

Rated delivery, m³,hr 60
Water head, m 20
Suction head, m 5
Motor rating, kW 5.5
Motor speed (synchron.), rpm 1500
Main dimensions, mm
    length 487
    width 397
    height 439
Weight of pump, kg 68.5

The equipment most suitable for an automatically controlled pumping installation is the НДВ (NDV) pump and submersible motor unit. The pump in this unit can operate whether wholly or partially submerged in water and does not require priming. The motor is of normal mining enclosure design and its normal working position is with the shaft vertical Fig. 150 shows the design features of the НДВ

Table 11

Specifications of Pump and Submersible Motor Unit

Characteristics	Pump unit			
	НДВ-6 (NDV-6)	НДВ-10 (NDV-10)	НДВ-20 (NDV-20)	НДВ 40 (NDV-40)
Rated delivery, m³/hr	6	10	20	40
Water head, m	8	17	17	17
Power rating, kW	0.55	1.5	2.3	4.5
Voltage, V	380	380	380	380
Rated current, A	2.1	4.2	6	10
Speed, rpm	2740	2740	2870	2850
Dimensions, mm				
height	455	475	560	640
length	310	365	370	410
width	310	365	320	365
Weight, kg	69	72	100	133

(NDV) pump unit and how it is mounted in a mine. The unit consists essentially of a motor *1* and a pump, within the casing *2* of which an impeller *3* rotates. A bolt *4* holds the impeller fixed on the motor shaft. At the bottom the pump is fitted with an inlet footpiece *5*. Bolts *6* join the pump to the motor. The motor is made watertight by enclosing it in a stainless steel thin-walled frame filled with MK-1 or MK-2 insulating compound.

The specifications of these pump and submersible motor units may be seen in Table 11.

Centrifugal Pump Troubles, Their Causes and Remedies

Trouble	Cause	Remedy
Pump delivery falls off	Fouled foot valve and intake screen	Clean foot valve and its screen. If compressed air line is available, foot valve and screen may be cleaned without removal by blow out with compressed air
	Air leakage in the suction line	Check suction line and pull flange joints tight
Pump fails to deliver water after start-up	Pump insufficiently primed before start	Shut down pump, prime it, and start it up again
Pump fails to develop full head	Motor speed is low	Check speed of motor
Motor heats due to overload	Thrust bearing is worn and impellers rub against casing, shaft journals seized in bearings	Re-babbitt bearing shells. Repair pump
Pump casing vibrates during operation	Motor and pump are improperly mounted	Tighten bolts fixing pump and motor to their support, level and align pump and motor shafts. Check rubber rings in flexible coupling and replace bad rings

J. FANS FOR BOOSTER VENTILATION

The main fan system is usually unable to ventilate adequately blind ends in mines, for example, entry headings during drivage operations. Therefore, it becomes necessary to put in booster or auxiliary fans.

The fans used for auxiliary and booster ventilation may be either centrifugal or axial-flow. At the present time the axial-flow (propeller) fan is the one mainly used for auxiliary ventilation.

Fig. 151 gives a general view of a centrifugal fan with a drum-shaped runner. The fan consists of a volute casing *1* in which a multibladed runner or wheel *2* runs. Air taken in by the fan through the central opening *3* is thrown out at the periphery of the runner into the casing and is directed into ventilating ducts by a diffuser *4*. Since a certain degree of vacuum is produced by the runner blading, a continuous flow of air is maintained.

The air will thus enter a centrifugal fan axially on either one or both sides and be discharged
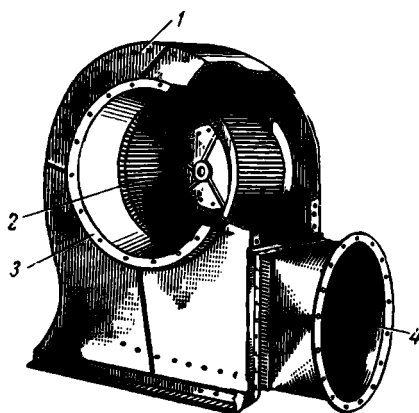
Fig. 151. Centrifugal fan with a drum-shaped runner
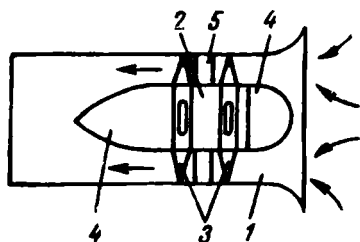
Fig. 152. Essential elements of an axial-flow fan

through the diffuser at right angles to it. The function of the diffuser is to reduce the velocity of the air as it leaves the runner blading.

Fundamentally, an axial-flow (propeller) fan (Fig. 152) consists of a casing *1* flanged at one or both ends for attachment of the air ducting, a built-in electric motor *2* with fan wheels *3* fitted on its shaft, the forward and rear fairings *4*, and a set of guide vanes *5* welded between the casing and the motor frame. For convenience and portability in mining conditions, auxiliary axial fan units are fitted with skids. Such a unit, when electric-driven, will have a built-in cable entrance box for connection of the supply cable.

A fan wheel consists of a hub carrying a set of propeller-shaped blades and is keyed to the motor shaft.

Each blade is attached to the hub at a definite angle and a hub may carry from 9 to 16 blades. The greater the number of blades, the greater the head developed by the fan.

Fig. 153 gives a general view of the ВП-4 (VP-4) axial-flow compressed-air drive fan.
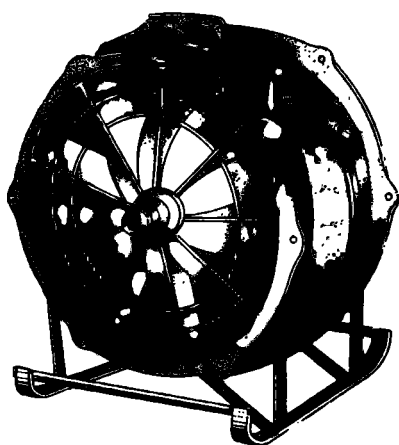
Fig. 153. BΠ-4 (VP-4) pneumatic
axial-flow fan

The Prokhodka-500 fan shown in Fig. 154 is a two-stage, high-head unit with a cylindrical casing in the centre of which a flameproof electric motor is housed. Each of the two motor shaft extensions carry a cast fan wheel. A set of stationary guide vanes is provided be-
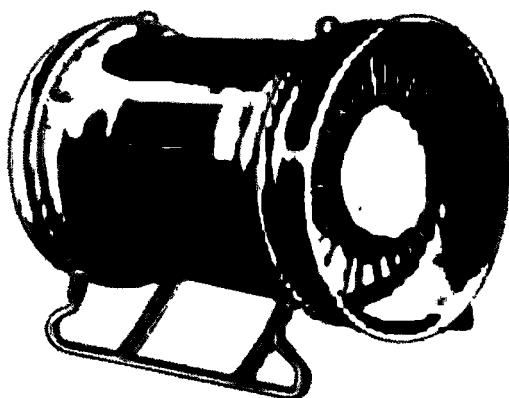
Fig. 154. Prokhodka-500 axial-flow fan

tween the fan wheels to guide and straighten the air flow from the first stage before it enters the second stage. The motor frame is welded to the guide vanes.

Ventilating Ducting

For booster and auxiliary ventilation, use is made of both metal and flexible (rubberized fabric, etc.) ducting.

Metal ventilating ducts are joined together by two methods. The first method is to fit each duct section with either one or two flanges for jointing with bolts. The second method is to make the duct sections for spigot joining, by which the conical end of one section fits into the mating expanded end of the next section. When the joints are made up, the mating surfaces should be coated with a sealing cement to avoid air leakage.

Rubberized-fabric ducting sections are very convenient to join by means of self-expanding steel rings. In making a joint, steel self-expanding rings are inserted in each end of the ducting to be joined, the ring on one end is compressed by hand, and the tubing is pushed into the other end so that its ring goes just beyond the ring in the outer duct end. Then, the inner ducting end and its ring are allowed to expand, and both ends drawn apart to make the rings approach and close the joint. The joint is completed by attaching a grooved clamping ring round the outside of the ducting at the two inner rings and fixing it with its hinged locking attachment.

When Prokhodka-500 fan units are employed in conjunction with Type M rubberized-fabric ducting it is possible to:

(1) Reliably ventilate a working of up to 500 m length with one fan;

(2) Ventilate particularly long blind workings with fans arranged in tandem;

(3) Reduce the time for ventilating a face after shot firing to a period of 10 to 15 minutes;

(4) Ventilate several faces with one fan.

Chapter VI

ELEMENTARY RADIO ELECTRONICS

6-1. The Oscillatory or Tuned Circuit

Every radio receiver or transmitter contains one or more oscillatory circuits round which it is built. Such a circuit consists of an inductance (a coil) and a capacitance (a capacitor) connected in series to form a closed LC circuit.

When an LC circuit is connected to a source of an alternating current, an emf of self-induction is generated in the coil.

As will be recalled*, an alternating current flowing in a conductor produces an alternating magnetic field around it. In turn, a change in the number of lines of magnetic force which cut the conductor induce an emf of self-induction in it. The magnitude of this emf depends on the rate of change of the magnetic flux linking with the conductor, i.e., on the rate of change of the circuit, current and also on a property known as inductance.

The unit of inductance is the henry (H). A circuit is said to have an inductance of one henry when a current change of one ampere per second induces an emf of self-induction of one volt. One thousandth of a henry is a millihenry (mH).

A straight conductor also has self-inductance when it carries a varying current, but it is very small in magnitude. If this same conductor is wound into a coil, the magnetic flux produced by the current flowing in it will then link with all the coil turns and lead to a significant rise in the inductance. To obtain large values of inductance, a core of steel or some other highly magnetic material is placed within the coil (such materials are known as ferromagnetic from the Latin "ferrum" for iron). The core makes it possible to effectively increase

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* See Section 3-13.

the flux linkages with the coil and thereby make it have a greater inductance. If it is necessary to obtain low inductances (of the order of several millihenries), the coils are wound without cores or wound on powdered-iron cores which have a low permeability. Coils are wound on powdered-iron cores when they have to operate at high-frequencies; solid or laminated steel cores are impermissible in such cases because of the large energy loss in the core due to alternating magnetization.

Powdered-iron cores consist of fine grains of a magnetic material, such as carbonyl iron, Alsifer, etc., mixed with a suitable bonding material (resins or waterglass) and hot pressed into core shapes at elevated temperatures.

The simplest form of inductance coil is one wound in a single cylindrical layer on a round bobbin of pressboard. For higher inductance, the coil is wound with several layers of wire in a special manner known as "universal".

The second component of an LC oscillatory circuit is the capacitor or condenser. In its simplest form a capacitor consists of two metal plates, or electrodes, separated by a layer of insulation, called the dielectric (which may be air). When a capacitor is connected to the terminals of a battery, it becomes charged; the plate connected to the positive pole accumulates a positive charge, while the plate connected to the negative pole accumulates a negative charge. The potential difference equal to the battery emf is produced across the plates and an electric field is set up in the dielectric. When the charged capacitor is disconnected from the battery, the energy stored in its electric field is retained and the capacitor thus becomes a source of electric energy. The larger the area of the plates, the larger the charge it will take before its potential difference is equal to the emf of the source. The capacity or capacitance therefore depends on the plate area.

The unit of capacitance is the farad (F), or the capacitance of a capacitor in which a charge of one coulomb of electricity produces a change of one volt in the potential difference between its plates. The farad is, however, much too large for practical work. Capacitance is usually measured in microfarads (μF). The microfarad is one-millionth of a farad. A still smaller unit is the micromicrofarad, or picofarad, which is one-millionth of a microfarad.

When a capacitor is connected in a circuit with an unvarying emf (a d.c. circuit), the current charging it will only flow around the circuit until the capacitor is charged to the full emf of the source. In a circuit where the emf is continuously varying in magnitude and reversing in direction (an a.c. circuit), on the other hand, the capacitor will be charged and discharged at the frequency of the applied emf, and an alternating current will flow around circuit all the time.

This is the reason why a capacitor will pass an alternating current
and block the way for a direct current.

In accordance with the dielectric employed in them, capacitors
are classed as air, mica, paper and ceramic capacitors. As to design,
capacitors may be fixed and variable. In the first group are those the
capacitance of which cannot be changed, while the second group in-
cludes those in which the capacitance can be varied at will over a
large range. Fixed capacitors are most frequently made in the form
of two long, metal-foil ribbons separated by a paper tape which have
been coiled together into a cylindrical roll. After leads have been
attached to the two foil ribbon edges, the capacitor is completed by
encasing it in a paper sleeve.

Variable capacitors consist of a series of interleaving movable
and fixed metal plates usually separated by air. The movable plates
are assembled on a common shaft and can enter the gaps between the
fixed plates. When all the movable plates are turned fully in between
the fixed plates, the capacitor has its maximum capacitance.

When a charged capacitor is connected to an induction coil, the
capacitor first discharges, and the energy of its electric field gradually
decreases. This energy, however, is not lost because the resultant dis-
charge current rises to certain maximum and produces a magnetic
field around the induction coil. Thus, on discharge, the energy of the
electric field in the capacitor is transformed into the energy of the
magnetic field around the induction coil. When the capacitor is fully
discharged, the voltage across its plates is zero. Now the magnetic
field of the induction coil begins to collapse, thereby inducing an emf
of self-induction which causes a flow of current which recharges the
capacitor. This current flows in the opposite direction so that the
positive plate now receives a negative charge and the negative plate
a positive charge. When the charge reaches its maximum value the
capacitor discharges again, this time in the opposite direction, and
the same process repeats itself.

Thus, a circuit consisting of an induction coil and a capacitor is
capable of supporting an alternating flow of charges, i.e., a flow of
current in opposite directions. Such alternations of charges are called
electrical oscillations, for which reason the circuit is referred to as an
oscillatory circuit. An LC oscillatory circuit can only produce damped
oscillations, which are oscillations whose amplitude decrease with
time. After a charged capacitor has been connected to an induction
coil, the electrical oscillations gradually stop or decay. This occurs
because part of the energy is expended during each oscillation in
overcoming the resistance of the conductors and is evolved as heat.
Every new oscillation thus causes a loss in the energy stored in the
circuit until all the energy is expended and ends in decay of the oscil-
lations.

The period, or frequency, of oscillations depends upon the capacitance and the inductance of the LC circuit. A capacitor of low capacitance takes less time to charge fully. On the other hand, in a low-inductance circuit the current will change at a higher rate. Therefore, as the capacitance and inductance of a circuit decrease, the period of its oscillations become shorter and the frequency rises. By suitably adjusting the values of the capacitance or inductance, a circuit can be adjusted (or tuned) to oscillate at the requisite frequency (hence the name "tuned circuit"). This is what is done in a radio receiver. When the tuning knob is turned, the capacitance of the variable capacitor is changed, and so is the natural frequency of the tuned circuit.

The oscillatory circuit is supplied with power (energy) from an external source. For this, an a.c. emf of definite frequency is applied to the circuit.

The oscillations which occur within a circuit due to some external source of energy are known as forced oscillations. If the frequency of the forced oscillation and the natural frequency of the circuit coincide, what is known as resonance is observed. During resonance the strength (or amplitude) of the oscillations in the circuit sharply increase. The effect of resonance can be obtained by means of a pendulum. By giving a pendulum slight pushes at the end of each swing in step with its natural period of oscillation, it can be made to fluctuate very widely by application of only a very slight force. An electrical circuit can also be made to "swing" or oscillate with great magnitude by feeding in the energy at the natural period of oscillation. This rise in amplitude, however, will only be observed when the applied frequency is the same as the natural frequency of the circuit. In other words, of all the frequencies applied, the circuit will only respond to the one corresponding to its own frequency. In this way the circuit can build up a considerable amount of energy received in very small portions from some external source.

6-2. Thermionic Valves

Thermionic valves (or tubes) depend for their operation on thermionic emission, i.e., the emission of free electrons by heated bodies. Thermionic valves are made in a great variety of types differing in application, performance, internal arrangements, size and form. However, they may be roughly divided into two groups: valves which rectify an alternating current, and valves which either amplify or generate signals. Valves in the first group have two electrodes and are called diodes; valves in the second have no less than three electrodes.

A diode consists of a glass or metal bulb from which the air has been evacuated, and two electrodes, one in the form of a tungsten filament

called the cathode, the other in the form of a metal cylinder called the anode (or plate) (Fig. 155). Both the anode and cathode have leads brought out through the base.

When a current is passed through the cathode, it heats the filament to a very high temperature (2000° to 2500°C) at which it begins to emit electrons.

If the anode is connected to the positive terminal of a source of current, the electrons, being negative, will be attracted by the anode, and a current will flow through the valve. If the anode is connected to the negative terminal of the current source, the flow of current through the valve will cease, because the electrons emitted by the filament will be repelled
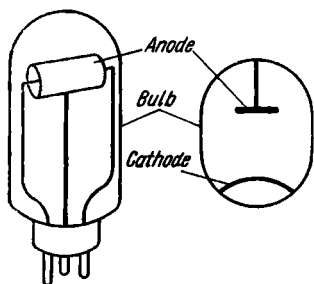
Fig. 155. Sketch and circuit diagram of a diode
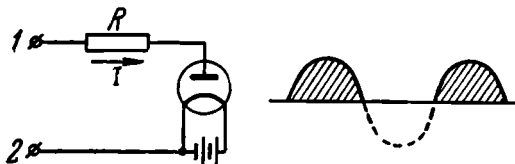
Fig. 156. Rectifier circuit diagram

by the anode. The electrons are therefore able to pass through a diode only in one direction, from the cathode to the anode, when the latter is positive with respect to the cathode. This is how a two-electrode valve acts as a rectifier. Fig. 156 shows a simple form of valve rectifier circuit for supply of a load R.

If at any given instant of time terminal *1* and, consequently, the anode are made positive, a current will flow through the load in the direction indicated by the arrow. One half-cycle later, when the polarity of terminals *1* and *2* reverses, the anode becomes negative, and the current flow ceases. The current thus passes through the load only during one half of the cycle (see right-hand part of Fig. 156).

An amplifying type of valve differs from a rectifying valve in that it is provided with a third electrode called the control grid, or simply the grid. Fig. 157 shows a three-electrode valve, called a triode, and the circuit into which it may be connected. The cathode in the triode is a filament, while the anode is a metal cylinder. The grid, made as a wire spiral, is between the cathode and the anode. When the grid is at a positive potential, it aids the anode in attracting the electrons, and so the flow of current through the valve increases.

If the grid is negative, it will repel the electrons, thereby limiting the flow of current through the valve. When the negative potential at the grid reaches a certain value, it completely stops the flow of

current through the valve, and the valve is said to be cut off, blocked.

Since the grid is placed closer to the cathode than to the anode, it has a much greater effect upon the electron flow than the plate. By applying a very small voltage to the grid it is possible to effect control over the valve current within a broad range.

In present-day valves (diodes and triodes) the cathode is very often not a filament, but a small hollow cylinder with a suitably coated
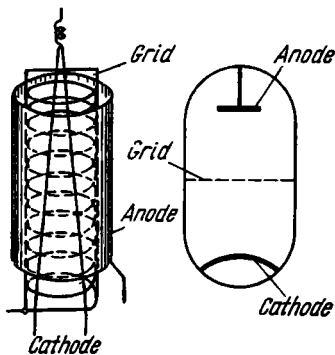
Fig. 157. Sketch and circuit diagram of a triode
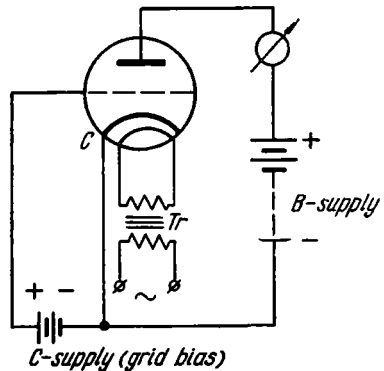
Fig. 158. Connection of an indirectly-heated triode

outer surface. The filament, or heater, is arranged within the cylinder. The surface of the cylinder from which the electrons are emitted is the actual cathode and is called indirectly heated, since the current passes through the heater. Valves with indirectly heated cathodes are generally supplied with a.c. by a step-down transformer (or a special heater winding on a power transformer). Fig. 158 shows an indirectly-heated triode connected in a circuit. The valve plate is maintained positive by an anode battery B (called the B supply) and the grid is made also positive by a grid battery C (known as the C supply), and so helps maintain a flow of current through the valve. The heater supply is provided by a step-down transformer Tr. Note that the valve cathode is connected to the negative terminals of both batteries.

Fig. 159 shows an elementary voltage amplifier circuit based on a triode. Supply for the heater is not shown in the figure. The anode circuit contains a load resistance R_A from across which the output voltage is taken off. When an input a.c. of small amplitude (input voltage V_{in}) is applied to the grid, the latter will be alternately made negative and positive. The anode current will consequently change in value, increasing when the grid is positive and decreasing when the

grid is negative, in step with the change occurring in the grid voltage V_{in}. Since small changes in grid voltage bring about considerable changes in the magnitude of the anode current, the amplitude of the latter will increase significantly.

Flowng through the load resistance R_A, the anode current I_A produces a voltage drop across it equal to $V_{out}=I_A R_A$. Since the anode current varies in step with the change occurring in the grid voltage, the output voltage varies in the same manner, but the amplitude of
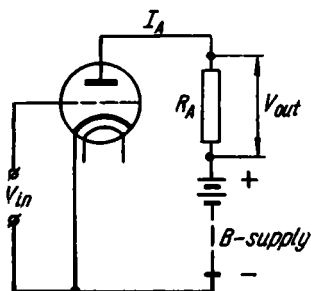
Fig. 159. Diagram of an elementary amplifier
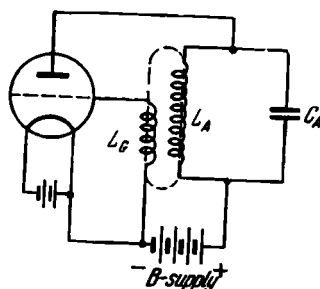
Fig. 160. Oscillatory circuit

the output voltage, however, is much greater than that of the grid voltage. Assume, for example, that the voltage applied to the grid changes from $+1.5$ V to -1.5 V and causes the anode current to change from 10 mA to 60 mA*. Let the load resistance $R_A = 6000$ ohms. The increase in the output voltage in comparison with the input voltage can be found as follows.

The input voltage changes between $+1.5$ V and -1.5 V, i.e., by 3 volts, producing a fluctuation current of $16-10=6$ mA in the anode current. The change in the voltage drop across the load resistance will therefore be:

$$V_{out}=\frac{16}{1000}\times 6000 - \frac{10}{1000}\times 6000 = 36 \text{ V}.$$

In the above calculation the anode current has to be expressed in amperes. From the above, we see that the output voltage is $\frac{36}{3}=12$ times greater than the input voltage. In other words, the thermionic valve has made it possible to obtain a voltage amplification of 12 times.

Where one valve is insufficient to give the required amplification for the operation of a device or relay, the amplified voltage of one

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* The anode current is small and is therefore measured in milliamperes.

valve can be fed to the grid of a second valve, and from the second valve to a third valve, and so on. In this way, cascade or multistage amplification can be accomplished.

In 1913 it was discovered that a triode could be used to obtain high-frequency electrical oscillations. Fig. 160 shows the circuit arrangement of such an oscillator or generator of electrical oscillations. The oscillatory circuit L_A and C_A is connected in the anode circuit of the valve. Next to the main inductance coil L_A is arranged an inductance coil L_G which is connected in series with the grid. The inductance coils L_A and L_G thus form the two windings of an air-core transformer. When the supply is applied to the valve, oscillations arise in the circuit, the direction of the current changes periodically, and so does the magnetic field around the inductance coil L_A. The alternating magnetic field will then induce an alternating emf of the same frequency in the grid coil. This emf makes the grid alternately positive and negative. As a result, the anode current will vary in step. In passing through the inductance coil L_A, the varying anode current will produce an additional voltage drop across the coil, thereby maintaining the oscillations of the circuit.

The $L_A C_A$ circuit maintains its oscillations owing to the energy supplied by the anode battery which supports the amplification of the grid voltage; the latter, in turn, makes up for the losses in the oscillatory circuit.

If one more inductance (aerial) coil is placed near the inductance coil L_A and connected to the aerial and earth, electromagnetic oscillations at the frequency of the oscillatory circuit will be induced in it. The resulting electromagnetic oscillations can then give up some of their energy in the form of electromagnetic waves radiated by the aerial into space. Such an arrangement is called a valve oscillator, and the oscillations generated by it are referred to as a carrier wave or frequency. The higher the frequency of such oscillations the greater the percentage of energy radiated by the transmitter.

6-3. Radio Transmission and Reception

The electromagnetic oscillations radiated by an aerial travel through space in all directions as do waves similar to those which appear when a stone is dropped into the water. Electromagnetic waves propagate at a speed equal to that of light, i.e., 300,000 kilometres per second.

Electromagnetic waves differ in frequency (cycles per second). While travelling at the same speed, waves of different frequency cover a different distance during each period of propagation and are

said to be of different wavelength.* If, for example, the frequency is 100,000 cycles per second, at the speed of propagation of 300,000 km/sec the 100,000 waves will cover a distance of 300,000 kilometres. The distance occupied by one wave or cycle, that is, the wavelength, will be $\frac{300,000}{100,000}$ =3 km, or 3000 m. Thus, wavelength decreases with increasing frequency.
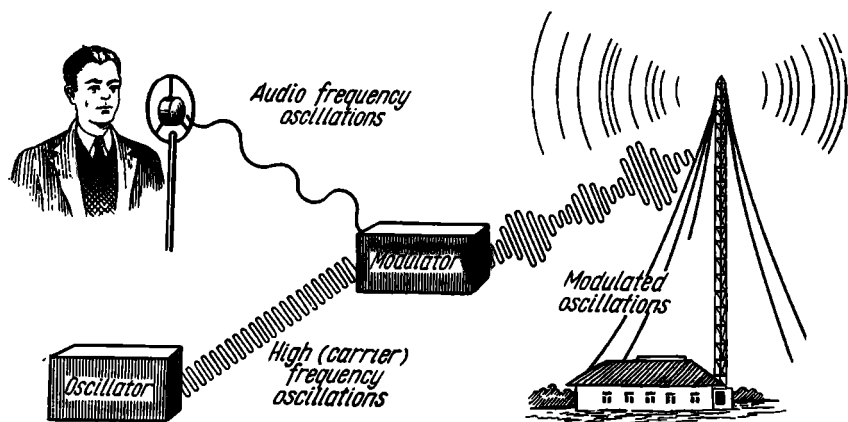
Fig. 161. Schematic representation of radio broadcasting

The electromagnetic oscillations employed in radio are conventionally divided according to their wavelengths into the following bands:

(1) long waves—from 2000 to 750 metres;
(2) medium waves—from 600 to 180 metres;
(3) short waves—from 80 to 10 metres;
(4) ultra-short waves—from 10 to 5 metres.

The above wavelength intervals are known as the radio broadcast bands. Decimetre and centimetre wavelengths are used for radar and radio navigation.** Millimetre wavelengths are used in radar and physics research.

Radio signals are transmitted and received as follows. At a radio transmitting station the high-frequency oscillations (carrier frequency) generated by the valve oscillator are fed into a unit termed the modulator (Fig. 161). Simultaneously, the modulator is fed

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* The wavelength may be defined as the distance between the peaks or "valleys".

** Radio navigation is the use of radio for determining the position and course of ships and aircraft.

with the electrical oscillations which are produced by voice or music in a microphone* and are an exact electrical replica of the sound. In the modulator, the audio (low) frequency signal from the microphone is superimposed upon the carrier (high) frequency so that the high-frequency oscillations emerging from the modulator carry the audio-frequency signal. In this form they are fed into the transmitting aerial for radiation into space in all directions.
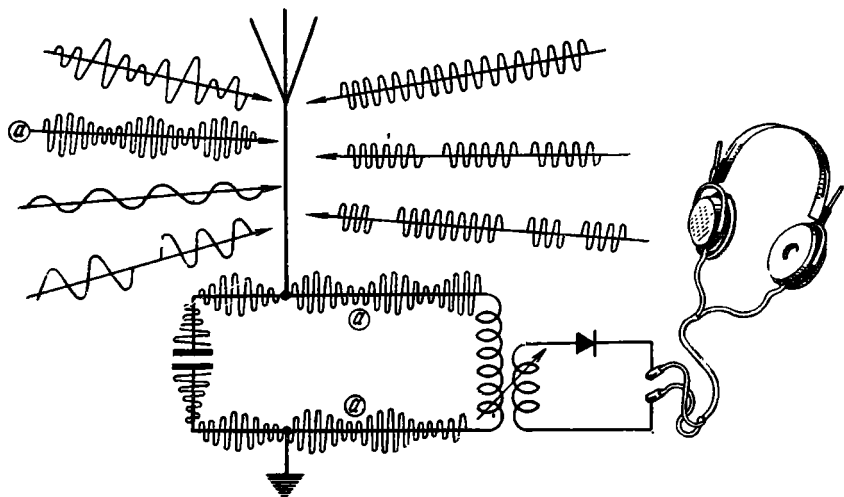
Fig. 162. Schematic diagram of radio reception

At the receiving end the aerial will simultaneously pick up waves from all directions. From them the desired frequency is selected and all other are suppressed. This is done by tuning the oscillatory or tuned circuit of the receiver to resonance with the desired frequency, or the wanted station. All the other frequencies are bypassed to earth through the earthing lead (Fig. 162). However, the oscillations selected by the tuned circuit are not intelligible to the human ear, and therefore require a "translator". This function in a radio receiver is performed by the detector. The latter converts the modulated carrier into an audio (sound) frequency capable of operating a telephone receiver. The latter consists of an electromagnet and

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* A microphone is a device for converting sound into electrical oscillations. The simplest microphone consists of a thin metal or carbon membrane placed in contact with carbon powder enclosed in a small capsule. When the microphone is spoken into, the microphone membrane transmits the sound vibrations to the carbon particles, causing the resistance of the carbon-layer to closely follow the changes in sound. This is why the current passing through the microphone faithfully reproduces the sound oscillations.

a thin steel membrane. When the winding on the magnetic core is energized with the audio (sound) frequency,* the magnetic field of the core makes the membrane vibrate in step with the current variation in the winding to alternately lower and raise the air pressure in front of it, in other words, makes it produce sound waves.
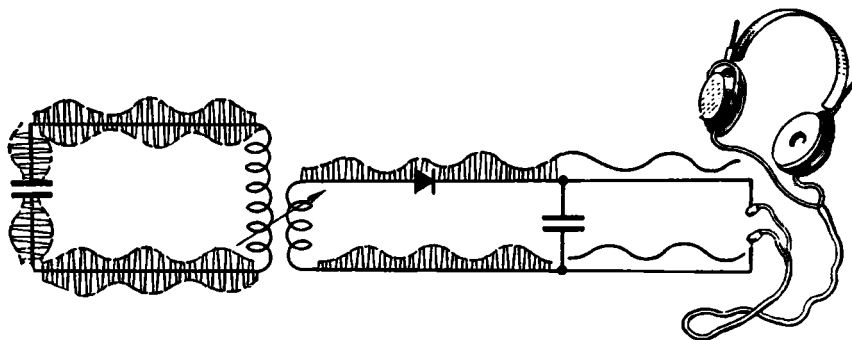
Fig. 163. Schematic representation of detector operation

A telephone receiver is only able to convert oscillations of relatively low frequency into sound waves. For high-frequency oscillations the telephone receiver coils present an extremely high impedance; furthermore, the membrane, being of finite mass, is unable to vibrate in step with the high-frequency oscillations. Finally, even if a telephone receiver were able to reproduce such oscillations, it would be to no aim because their frequencies are beyond the limits of what the human ear can discern.

The simplest detector consists of a crystal of galena (zinc sulphide) and a steel spiral held in point contact with one of the faces of the crystal. This contact pair functions in the manner of a pump valve or rectifier, since it permits the alternating current to flow only in one direction, from the point contact into the face of the crystal, in the form of separate pulses of unidirectional current. The latter is what is called a pulsating current which can be fed into a telephone receiver (Fig. 163). In the telephone receivers this pulsating current is converted into sound perceptible by the ear.

Fig. 164 shows modulation and detection in graph form. During a broadcast, voice or music produces sound waves (graph *a*). They cause the microphone current to fluctuate (graph *b*). By enveloping the carrier, this varying microphone current, or the modulating

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* The audio-frequency band extends from 16 to 16,000 cycles per second. Only oscillations within this range will be heard by the human ear as sound.

signal, produces the modulated carrier which is then fed into the aerial. When the modulated carrier is fed into the detector of a receiver, the current pulses passing through the detector will vary
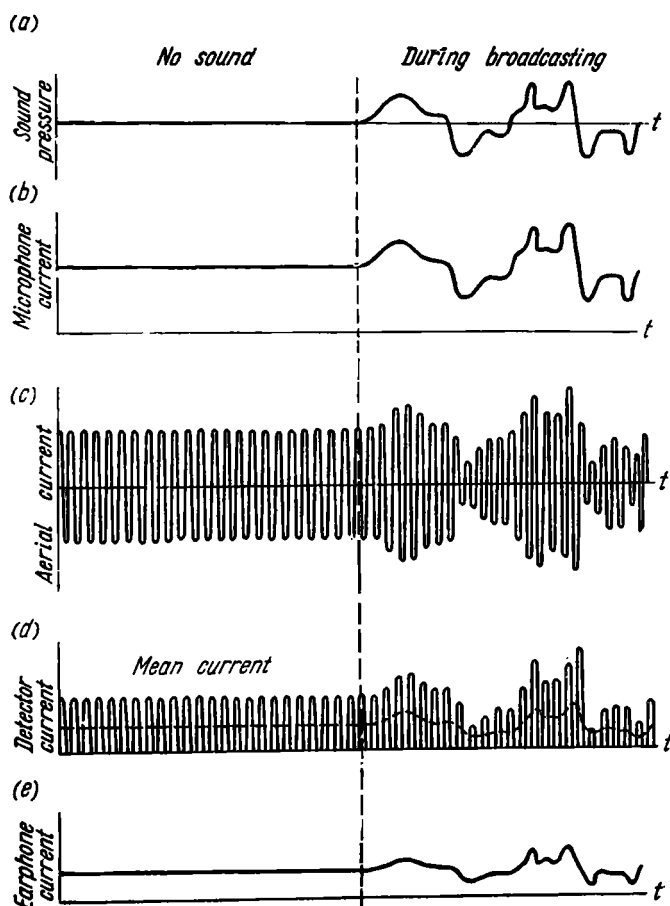
Fig. 164. Modulation and demodulation

with time, producing a current the mean value of which will change in step with the varying audio frequency (graph *d*)*. The waveform of this mean current will be an exact copy of the microphone current or the intelligence waveform (graph *e*).

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* The graph shows that the detector only passes upper halves of the oscillations.

From the detector, the audio frequency signal is passed through a telephone while the radio frequency currents are shunted round the telephone by a bypass capacitor.

The detector used in the modern radio receiver is a diode valve which is much more stable in operation than a crystal-point-contact pair. As in the case of the crystal arrangement, the current can only pass through the valve in one direction.

For raising the sensitivity of reception, the voltage from across the tuned circuit is applied to the grid of a triode valve to obtain an amplified output. Since even a small change in grid voltage produces a considerable increase in anode output, the valve in this case can drive several headphones or loudspeakers, which cannot be done by the weak current picked up by the aerial and tuned circuit. The triode valve thus performs the functions of an amplifier and of a detector at one time.

In recent years amplifier and detector valves have given way to semiconductor devices. "Semiconductor" is the term used to identify a material with an electrical conductivity between that of a conductor and insulator. Materials such as germanium, silicon, selenium, copper oxide, etc., are semiconductors. In conductors, the outer electrons are only loosely retained in their atoms and can freely travel within the conductor. Due to this the electrical conductivity is high. In an insulator, the outer electrons are strongly bound to their atoms; there are no free electrons to effect a flow of changes and an insulator is therefore unable to conduct current. In semiconductors most of the electrons are but loosely attached to their nuclei at low temperatures. Therefore, when a semiconductor is heated, its atoms are agitated to such a degree that they lose their outer electrons; the number of free electrons increases, and the conductivity rises. A conductor, when heated, on the contrary, loses some of its electrical conductivity.

Furthermore, in semiconductors charges are carried not only by the electrons driven loose from their atoms, but also by those which remain attached to their nuclei. Consider a piece of a semiconductor connected across the terminals of a battery. The electrons thrown off by the continuously oscillating atoms are caught up by the applied electric field and forced to move to the positive terminal of the battery. As they break loose from their atoms, the electrons leave an electron deficiency behind. The electric field, however, immediately causes an electron from an adjacent atom to fill the deficiency, leaving an electron deficiency in its atom and so on. The bound electrons thus move to fill the consecutive electron deficiencies in the atoms (or the "holes", as physicists call them) in the direction of the positive terminal of the battery. This behaviour noticeably increases the electrical conductivity of a semiconductor.

Semiconductors in which electrical conduction is mainly maintained by free electrons are called *n*-type semiconductors. Those in which electrical conductivity is due to "holes" are known as *p*-type semiconductors.

By adding a minute amount of certain substances called impurities, a semiconductor can be made a purely *n*-type or *p*-type semiconductor.

The semiconductor devices most widely used in place of valves are germanium diodes (rectifiers) and triodes. Germanium is a metallic element of a silvery-white colour. The addition of antimony, phosphorus, arsenic or tin (known as donor impurities) impart
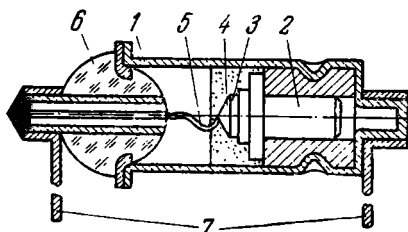
Fig. 165. Parts of a germanium diode

germanium *n*-conduction. The addition of indium or boron (called acceptor impurities) give it *p*-conduction. Fig. 165 shows the arrangement of a germanium diode. A metal case *1*, 7 mm in diameter and 20 mm long houses a metal base *2* which supports a crystal of germanium *3*, which makes one part of the junction. The other part of the junction is an indium wafer *4*. It is in contact with a cat-whisker *5* led out through a glass insulating bead *6*. At the ends, the diode is provided with a set of leads *7*. Such diodes are very small in size and mechanically strong. They have found a broad field of application owing to long service life, absence of heater circuits, and relative ease of manufacture. Fig. 166 schematically shows the parts and connections of a germanium triode, or point-contact transistor. A germanium crystal triode, or transistor is a germanium wafer having three electrodes attached to it. The electrode welded to the wafer is called the base. The other two electrodes or cat-whiskers, called the emitter and the collector, are placed very close together in point contact with the surface of the germanium wafer. The wafer possesses both *p*-type and *n*-type conduction due to the addition of respective impurities. At the base, the wafer is *n*-conductive, and at the catwhiskers it is *p*-conductive. A voltage is applied between the base and the collector by a battery the negative side of which is connected to the collector. A second battery is

connected between the base and the emitter in order to make the emitter positive. The voltage to be amplified is applied to the emitter and the amplified voltage is taken off from the collector.
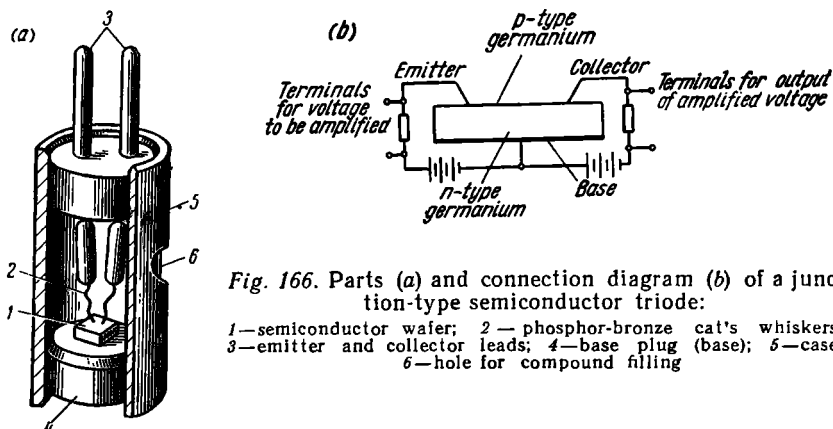
Fig. 166. Parts (a) and connection diagram (b) of a junction-type semiconductor triode:

1—semiconductor wafer; *2* — phosphor-bronze cat's whiskers; *3*—emitter and collector leads; *4*—base plug (base); *5*—case; *6*—hole for compound filling

To a certain degree the action of the transistor is similar to that of amplifier valve. The emitter corresponds to the cathode, the base to the grid, and the collector to the plate (anode). In the valve, an increase in positive grid potential leads to a rise in anode current. The corresponding rise in collector current is obtained by raising the voltage between the emitter and the base.

6-4. Photoelectric Cells

Photoelectric cells are devices in which incident radiation (commonly light) produces either (a) emission of electrons with the resultant flow of current (the outer photoelectric effect or the photoemissive effect); or (b) a change in electrical conductivity (the inner photoelectric effect or the photoconductive effect); or (c) generation of an emf or voltage (the photovoltaic effect).

Devices based on the photoemissive effect are called photoemissive cells, or simply photocells, or phototubes. Devices utilizing the photoconductive effect are termed photoconductive cells. Devices which depend for their operation on the photovoltaic effect are referred to as photovoltaic cells.

Photoemissive Cells. Fig. 167 shows the parts and circuit arrangement of a photoemissive cell or phototube. A glass bulb houses a ring-shaped metal anode connected to a pin in the base. The second electrode, the cathode, is a light-sensitive film of two chemical elements (antimony and cesium) deposited upon the internal sur-

face of the bulb. When light falls on the cathode, electrons are emitted from the cathode surface and are attracted by the positive anode. When the external circuit is closed, a current flows around it.

Photoemissive cells are made in vacuum and gas-filled types. In vacuum photoemissive cells all air is exhausted from the bulb in order to obtain a large photoelectric current. If air were present in the bulb, the electrons emitted by the cathode would collide with the air molecules and be dispersed. Owing to this, only a small number of electrons would be able to reach the anode, and the current in such a cell would be therefore very small.
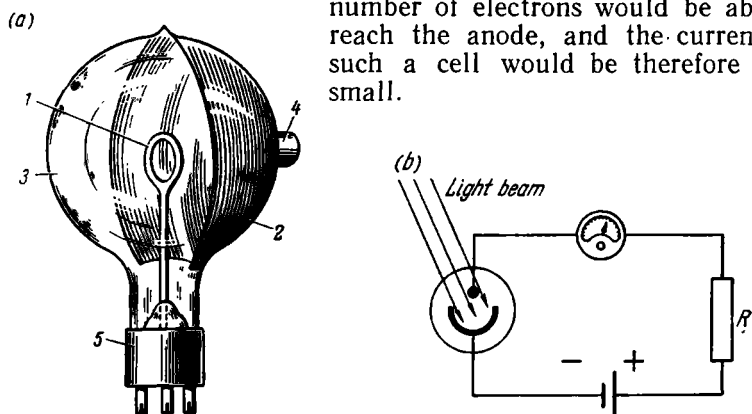
Fig. 167. Parts (*a*) and connection diagram (*b*) of a photoemissive cell:
1—anode; *2*—cathode; *3*—glass bulb; *4*—cathode lead; *5*—base

The gas-filled photoemissive cell has its bulb filled with inert argon gas. In this case the electrons emitted by the cathode collide at high speeds with the argon molecules in their travel towards the anode and knock more electrons out of them. These electrons also travel towards the anode to augment the flow of electrons through the cell. Since the gas molecules which lost their electrons become positive ions, they travel to the negatively charged cathode. In striking the cathode, they cause it to emit still more electrons, and so on. At the cathode the positive ions receive the lacking electrons and become neutral gas molecules. The photoelectric current in a gas-filled photoemissive cell therefore consists of two components; one due to the direct action of light, the other, and by far the larger, due to the ionization of the gas.

Fig. 168 shows a photoelectric relay circuit employing a photo-emissive cell. A valve amplifies the photoelectric current.

When light from a lamp source *1* strikes the cathode of the photocell *2*, current I_{ph} flows through the cell in the direction indicated by the arrows. This current flow produces a voltage drop $V_g = I_{ph}R_g$

10*

across the grid resistor R_g. Since a current always flows from a point of higher potential to one of lower potential, the upper end of the resistor R_g and, consequently, the grid of the amplifying valve *3* are at a negative potential relative to the lower end of the resistor R_g and the valve cathode. The valve is thus cut off and no anode current flows through it. If some object now interrupts the light
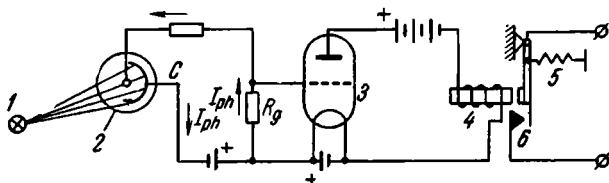
Fig. 168. Diagram of a photoelectric relay

beam between the source and the cell, current I_{ph} and voltage drop V_g disappear, the valve grid loses its negative potential, the valve becomes conducting and a current flows in its anode circuit. The anode current, by passage through the winding of an electromagnet
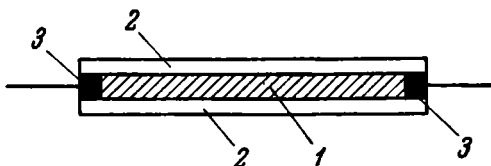
Fig. 169. Design of a photoconductive cell:
1—semiconductor wafer; *2*—quartz or glass plates; *3*—leads

4, magnetizes the core to pick up the armature *5* and close the contact *6* in a control circuit. Photoelectric relays of this type serve to automatically count articles travelling on conveyors, protect against ingress to dangerous zones, control the level of a material in a hopper, etc.

Photoconductive cells are photosensitive devices the electrical resistance of which varies inversely with the intensity of light falling on it. This property is possessed by certain semiconductors. Fig. 169 shows the parts of a photoconductive cell. The layer of a semiconductor material *1* (selenium, thallium sulphide, lead sulphide) is deposited between two quartz or glass plates *2*. Connection into a circuit is by means of electrodes *3* attached to two opposite ends of the semiconductor layer. When the electrodes are connected to the terminals of a battery, the value of the current flowing around

the circuit will depend on the intensity of the light allowed to fall on the photoconductive cell.

Photovoltaic or barrier-layer cells are capable of operating without an external source; current can only pass through them in one direction.

The appearance and arrangement of such a cell is shown in Fig. 170. A semiconductor layer *1* is applied to a metal baseplate *2* and
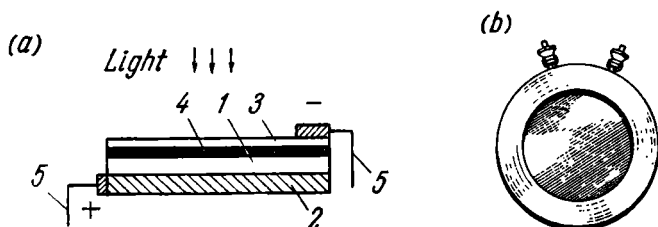
Fig. 170. Design (*a*) and external view (*b*) of a photovoltaic cell

then a thin film of transparent metal *3* is sprayed over the semiconductor layer. After appropriate heat treatment, a film *4* is produced between the semiconductor layer and its transparent coating. This film, called the barrier layer, is only capable of passing electrons in one direction, from the semiconductor to the metal. When light passes through the transparent metal and falls on the semiconductor layer, electrons are liberated which pass across the barrier layer, developing a potential difference across the cell electrodes *5* by means of which a current can be made to flow in an external circuit.

Chapter VII

POWER DISTRIBUTION IN MINES

A. MINE POWER SUPPLY CIRCUITS

Mines in the U.S.S.R. generally receive electric power from area authority substations at a voltage of 6000 volts. The power is received, transformed and distributed by a substation built on the territory of the mine. From it, outgoing high-voltage feeders are run to the larger power loads of the mine (winders and ventilating fans) and also to a central underground substation. Such a system is employed where the seams lie at relatively great depths.

The central underground substation (Fig. 171) is supplied by two incoming feeder cables run down the wall of the mine shaft and is located in a chamber near the shaft. Each of its incoming and outgoing feeder cables is led into a separate high-voltage switchgear compartment or cubicle equipped with an isolator, oil circuit breaker and protective devices.

From the central underground substation the power is distributed to the districts. In view of their considerable distances from the substation, this is done at high voltages of 6 kV or 3 kV.

A district transformer substation usually consists of one or two power transformers, high-voltage switchgear units for the switching and protection of the transformers, and feeder air circuit breakers for switch-in, switch-out and protection of the outgoing feeders installed between the district transformer substation and the local distribution switchboards or centres. One such local board may supply power to the units of two or three working or heading faces.

In the Moscow coal fields and at some mines in the Kuznetsk coal fields, where the seams do not lie very deep, the electric motors of the district units are supplied from pole-mounted transformer
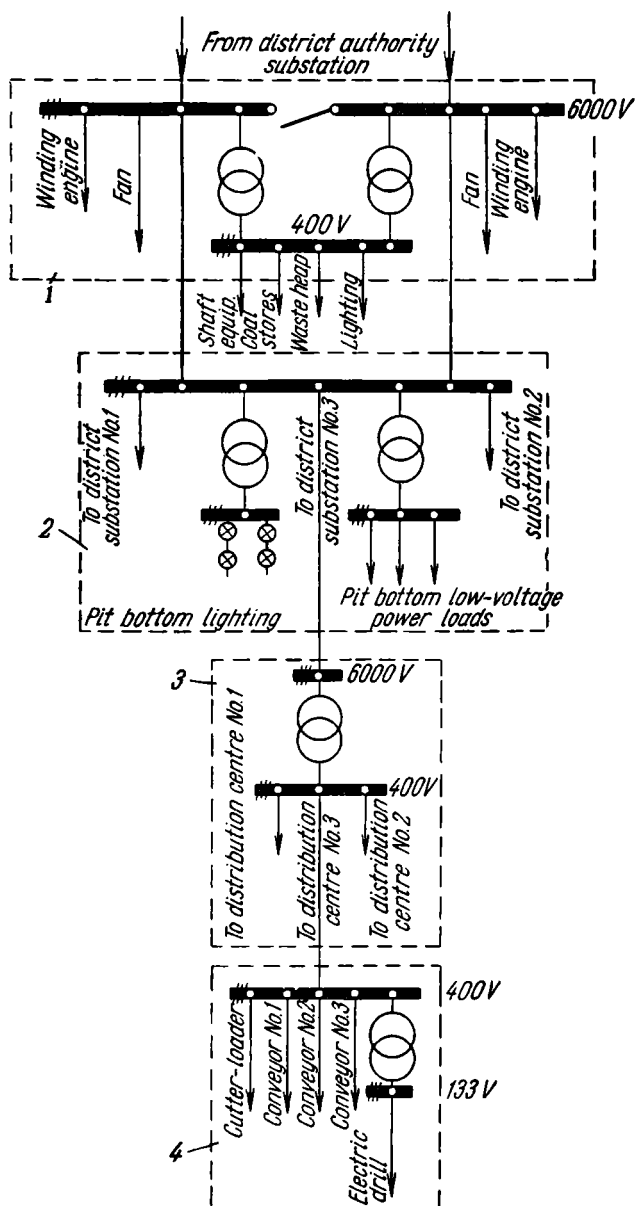
Fig. 171. Electric power distribution scheme:

1—surface main substation; *2*—central underground substation; *3*—district substation; *4*—gate-end distribution centre

substations on the surface by lowering the feed cables in special
encased bore holes. Such substations are situated near the cable
bore holes.

B. UNDERGROUND DISTRICT TRANSFORMER SUBSTATIONS

The function of a district transformer substation is to step down
the 6 or 3kV primary voltage to 400V and distribute the low-
voltage power to the various loads in its area.

Such a transformer substation is a very vital link in the general
mine power supply system, since uninterrupted operation of all
the units of mechanical equipment depends on continuity of supply.

These substations are installed in special rooms or chambers which
have to conform to the following safety rules:

(1) The chambers shall be faced with fireproof materials (con-
crete, brickwork, etc.);

(2) The chamber shall be ventilated so that the temperature
within it will not exceed the temperature of the surrounding work-
ings by more than 5°C, for which purpose two airways shall be
provided for intake and return of ventilating air (one air passage
to the gate road, the other to the rise heading);

(3) Both airways shall be fitted with latticed doors for free flow
of air and also with airtight fire doors which, when a fire breaks
out, can be closed to stop any inflow of air to the chamber. These
chamber doors shall open outwards and not interfere with the
movement of men or loads in the workings;

(4) The high-voltage and low-voltage switchgear shall be ar-
ranged at opposite ends of the substation chamber;

(5) The separate units of apparatus within the substation shall
be interconnected only by means of armoured cable.

A standard layout and connection scheme for a district trans-
former substation with two transformers can be seen in Fig. 172.

7-1. Mining Power Transformers

The ТМШ (TMSh) mining power transformer (Fig. 173) is built
into an oval-shaped ruggedized tank in which radiator tubes are
welded for better cooling of the transformer. The tank is filled with
transformer oil.

At opposite sides of the tank the transformer is fitted with cable
sealing boxes within which the cables are terminated and joint
sealed (on the older models of transformers the boxes were compound-
filled).

Fig. 172. Standard transformer substation with two 180 kVA power transformers: *a*—layout; *b*—main connections; *1*—ТМШ-180/6 transformer; *2*—УРВ-6/3 (URV-6/3) high-voltage switchgear unit; *3*—АФВ-1532 (AFV-1532) feeder air circuit breaker box; *4*—РУВ (RUV) earth leakage relay; *5*—ТСШ (TSSh) lighting transformer; *6*—ПВГ-380 (PVG-380) manual starter; *7*—lighting fitting; *8*—wire-rope pulley system; *9*—fusible fire protection links; *10*—3-mm diameter wire rope; *11*—foam-type fire extinguisher; *12*—dry-type fire extinguisher; *13*—sand box

To permit the oil to expand, the transformer has a free space left in the upper part of its tank. For oil filling, a hole *1* is provided at the top. It is closed with a screw plug. The oil level is indicated by an oil gauge *2* on which a mark is made to show the necessary level at 15° C. An oil drain outlet *3* is provided at the tank bottom for sampling or draining the oil. It is also closed with a screw plug.
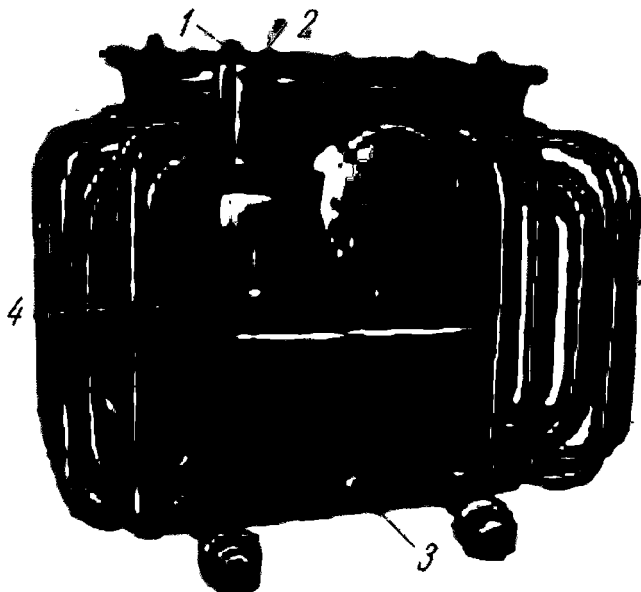
Fig. 173. TMШ (TMSh) mining transformer

Bolt *4* provides the earth connection for the transformer tank. The temperature of the oil is taken by a mercury-bulb thermometer conveniently arranged in a pocket in the cover.

The transformer can be moved on flanged wheels riding a rail track. The transformer has additional taps on the H. V. side to compensate up to ±5% deviations from rated supply voltage. When the supply voltage is below the rated value, connection is made to taps marked —5%. If the supply voltage is greater than rated value, the ⁺ 5% taps are used. The middle taps are used when the supply voltage is at its rated level.

Oil-immersed transformers are a source of great fire hazard. This hazard is obviated in the TCШВ (TSShV) dry-type mining power transformer by its flameproof design.

Such a transformer is much more difficult to cool than is an oil-immersed transformer. It is therefore designed to withstand a greater temperature rise. This is attained by using heat-resistant
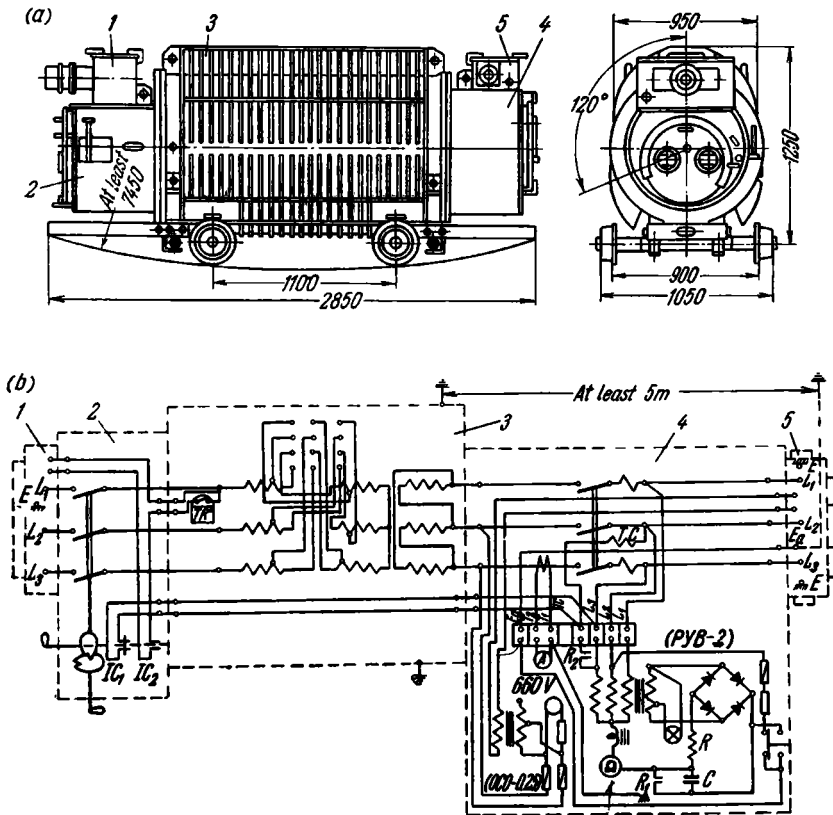
Fig. 174. Transportable mining substation:

a—general view; *b*—circuit diagram; *1*—high-voltage terminal box; *2*—high-voltage iso-lator box; *3*—dry-type power transformer; *4*—line air circuit breaker and earth leakage re-lay; *5*—low-voltage terminal box

insulation incorporating silicone impregnating varnishes and by providing the transformer casing with an adequate number of radiating ribs or fins.

The development of dry-type mining transformers has made it possible to build a transportable transformer substation having all its equipment mounted on a truck frame. Fig. 174 gives two general views and the circuit diagram of such a substation. These substations are intended for operation at voltages of 6000/400;

3000/400; 6000/693 and 3000/693 V. The above voltages are obtained by re-positioning the links or jumpers which bring the requisite number of primary turns into circuit.

The transportable substation comprises a ТСШВ-180/6 (TSShV-180/6) dry-type power transformer of 180 kVA rating, a three-pole isolator, a line circuit breaker box containing an air circuit breaker and a РУВ-2 (RUV-2) earth leakage relay, an ОСО-0.25 (OSO-0.25) lighting transformer, and indicating instruments. All the equipment is housed in flameproof enclosures. The temperature within the transformer enclosure is controlled by thermal relay TR.

The substation also incorporates mechanical and electrical interlocks. The mechanical interlock prevents the main cover of the line circuit breaker box from being opened when the isolator is connected to the high-voltage supply. The electrical interlocks prevent performing any operations with the substation isolator when the feeder circuit breaker in the central underground substation is closed, and also prevent it from being opened when the load is on, i. e., when the line circuit breaker on the low-voltage side is closed. The electric interlocks are provided by two contacts IC_1 and IC_2.

Power to these transportable substations is supplied through a flexible cable brought into a special sealing box. On the low-voltage side power is taken off by a ГРШН (GRShN) flexible cable of up to 70 mm² conductor size.

Transformer Mounting and Operation

Each transformer must be installed in the substation chamber according to the layout drawing. After it is put into its permanent position, the cables are connected and the tank or casing is earthed. Non-flameproof cable boxes must be filled with sealing compound as soon as the cable terminating work is completed. When the transformer has flameproof cable sealing boxes, only the sealing-end parts are filled with compound.

If the transformer has been lowered and installed without its oil being tested, a sample of the oil must be taken for chemical tests or a voltage breakdown test. Then the transformer is thoroughly inspected, and all loose bolts and screws drawn snug. If necessary, fresh dry oil is added. After this the transformer can be put into operation. At least once a shift the transformer should be inspected visually (without de-energizing it) for oil level, oil leakage from the tank or compound leakage from the cable boxes, overheating, humming in the transformer, continuity of the earthing, damage to the cables, etc.

If it is discovered that the oil level is low, an oil leak exists, compound leaks from a box, or the transformer hums abnormally, the

transformer must immediately be disconnected from the supply. A drop in oil level when the transformer carries a constant load is a sign of a leak in the tank; a rise in oil level means that the windings have become overheated because of an overload or an internal fault within the transformer.

A poor contact at the terminals of cable conductors may lead to compound leakage from the cable box. Abnormal hum in a transformer is an indication of a loose core clamping bolts or of a rise in the supply voltage.

When scheduled maintenance is done, the transformer should be disconnected from the supply and cleaned of dirt and dust. Then the insulation resistance of its windings should be measured, the contact joints and earthing bolts drawn tight where needed, and the insulators cleaned of dust and checked for possible damage. This routine inspection is also usually followed out on a district supply transformer after the district substation has been transferred to a new chamber.

At least once every six months a sample of the oil should be taken from the transformer for dielectric strength (breakdown voltage) tests. Not less than once in 12 months sludge and acidity tests should be performed.

7-2. Distribution Switchgear for Voltages over 1000V

High-voltage switchgear units and switchboards serve the following purposes:

(a) to receive and distribute electric power at a high voltage;

(b) to close and open electric circuits under normal operating conditions (so-called routine switching);

(c) to disconnect automatically a circuit section in the event of a short circuit, overload, or excessive drop in supply voltage;

(d) to disconnect branch circuit feeders for inspection and repair.

A high-voltage distribution centre is assembled of self-contained switchgear units or cubicles of the flameproof [УРВ-6/3 (URV-6/3), РВД-6/3 (RVD-6/3)] and normal mining [ВЯП-6 (VYaP-6)] types.

Flameproof switchgear units may be installed in all workings where the hazard of a gas and coal-dust explosion exists.

Normal mining type switchgear units are approved for installation in all workings of mines free from gas or coal-dust hazards and also in the central underground substations of gas or coal-dust hazardous mines.

Universal УРВ-6/3 (URV-6/3) switchgear units are intended to feed the loads at 6 and 3 kV and are available in current ratings of 20, 30, 50, 100, 150, 200 and 300 A.
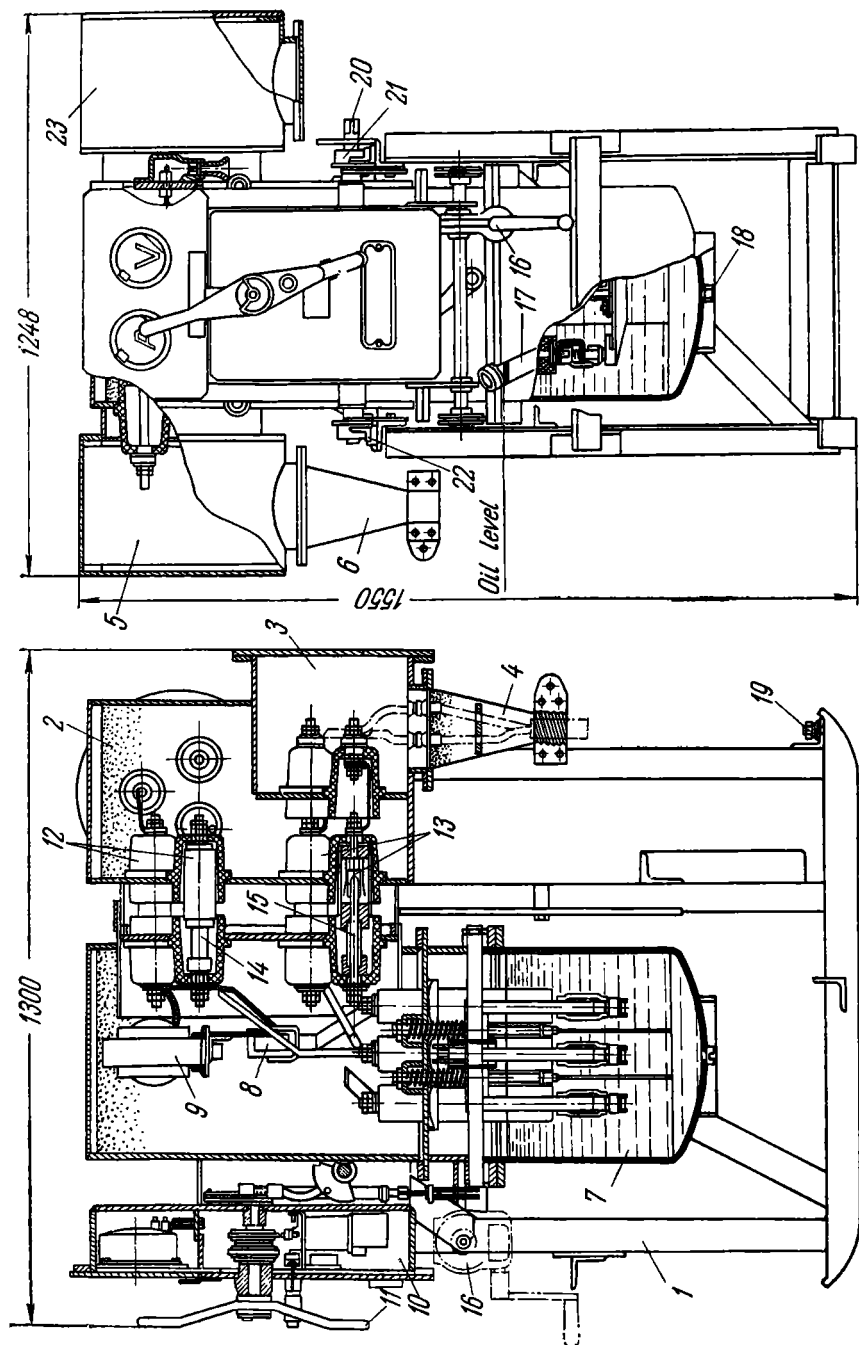
Fig. 175. Features of УРВ-6/3 (URW-6/3) flameproof switchgear unit

The URV-6/3 (Fig. 175) comprises a stationary assembly mounted on a pedestal framework *1* and a withdrawable part which can be drawn out in a horizontal direction.

The stationary part consists of a busbar compartment *2*, outgoing cable box *3* and its cable sealing end *4*, and an incoming cable box *5* with its cable sealing end *6*. The withdrawable part incorporates an oil circuit breaker *7*, two current transformers *8*, single-phase voltage transformer *9*, circuit-breaker operating mechanism *10* and its overcurrent and undervoltage relays, indicating instruments, and breaker closing handle *11*.

Both the busbar compartment and upper space of the withdrawal part containing the current and voltage transformers are filled with cable compound. After the incoming and outgoing cables are connected, the sealing-end boxes are also filled with compound.

The power circuit is made between the stationary and the withdrawal part through plug and socket isolators. In the stationary part are mounted the three upper isolator sockets *12* and the three lower isolator sockets *13*. Through connections made with the busbars, sockets *12* receive the supply from the incoming cable. The outgoing cable is connected to sockets *13*. The upper isolator plugs *14* are connected to the terminals on the incoming side of the oil circuit breaker through the current transformers, while the lower isolator plugs *15* make connection with the terminals on the outgoing side.

When the withdrawal part is moved out, only sockets *12* remain alive, all the other current-carrying parts of the unit are de-energized and can be inspected or worked on. A winch *16* is fitted to lower and raise the oil circuit breaker. An opening *18* closed by a plug is provided for draining and sampling the oil. The earthing connection is made at screw *19*.

When the withdrawal part must be moved out, a handle is placed on the end of shaft *20* and turned so that the part rides out on rollers *21* by running on guide rails *22*.

Single units of this type of switchgear are installed in district or inbye substations with one transformer, or to connect a high-voltage motor to the supply. When two or more units are required, they are joined together into a switchboard through the cable boxes *23* from which the face covers can be removed and the cable sealing-end fittings can be replaced by blind flanges. The busbars of the adjoining units are interconnected by suitable flexible conductors passed through the access openings in the cable boxes.

Fig. 176 gives the electric circuit of this unit. The bus isolating switch *BIS* and the oil circuit breaker *OCB* are used for routine switching operations in the power circuit. The overcurrent relays *OCR* and ammeter *A* carry current from the two current transformers

CT, while the voltage transformer VT energizes the coils of the undervoltage relay UVR and voltmeter V. The winding of the undervoltage relay UVR consists of two coils for connection in series with the 6-kV supply and in parallel with the 3-kV supply.
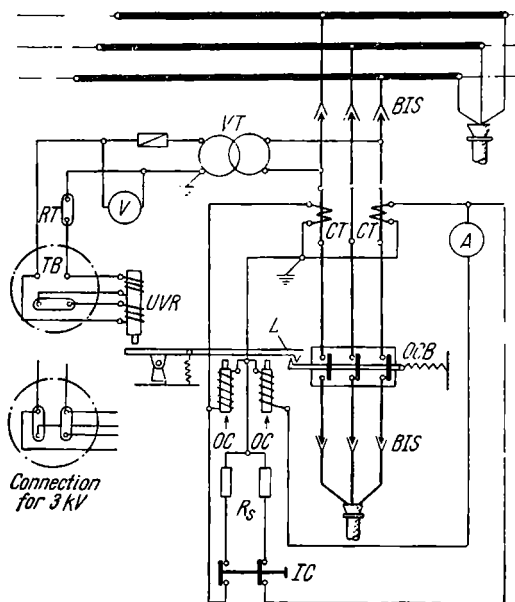
Fig. 176. Circuit diagram of УРВ-6/3 (URV-6/3)
switchgear unit

The above-mentioned connections are made on a link or jumper terminal block TB. Whenever any one of the relays operates, its core trips the latch L and the oil circuit breaker opens. During the starting of high-voltage motors (if the unit is used to control such motors), the overcurrent relays and the ammeter are shunted by a set of interlock contacts IC and shunting resistors R_s.

When the circuit breaker is open interlocks IC are closed, and the shunting resistors R_s are connected in series with the current transformers. The magnitude of the current passing through overload relays OC and the ammeter will therefore be lowered. The interlocks IC are linked with the circuit-breaker operating mechanism and open as soon as the circuit breaker fully closes.

The circuit breaker can be remotely opened by means of two terminals RT connected to a push button with normally-closed contacts. When the button is pressed, the coil of the undervoltage relay

UVR is de-energized, and the oil circuit breaker opens. Safety in operation of the cubicle is ensured as follows:

(1) the plug-and-socket isolators will not close or open when the oil circuit breaker is closed;

(2) the oil circuit breaker will not close if the withdrawable part is in any intermediate position;

(3) the cover of the operating mechanism will come off only after the withdrawable part is fully retracted;

(4) the withdrawable part will not travel inward (close) if the operating mechanism cover is removed or the oil circuit breaker tank is lowered;

(5) the oil circuit breaker tank can only be lowered after the withdrawable part is fully drawn out.

The УРВ-6 (URV-6) model of the earlier manufacture differs from the УРВ-6/3 (URV-6/3) in certain details of construction, and also in the absence of a shunting arrangement and means for change-over of the undervoltage relay windings from 6kV to 3kV.

The ВЯП-6 (VYaP-6) normal mining high-voltage switchgear units are manufactured for 6kV service not only in underground central substations, but also as self-contained units. They are available in the following modifications:

(1) for incoming feeders;

(2) for sectionalizing;

(3) for outgoing feeders;

(4) for motor starting;

(5) for single unit installation.

Each unit comprises a stationary and a withdrawable part. The withdrawable part (the truck) carries an oil circuit breaker, the blades of the top isolator on the front wall, the blades of the isolator on the rear wall, the oil circuit breaker operating mechanism, the upper-isolator operating mechanism, and the remaining associated apparatus (a voltage transformer, signal lamps, indicating instruments), depending on the modification of the unit.

The rear-wall-mounted isolator incorporates "floating" type contacts which will ensure a reliable connection when the truck is rolled in, even if the sockets and plug-in knives are out of line with each other. The isolator mounted on the front wall is of the conventional wedge-contact knife type.

The stationary part accommodates the socket side of the "floating" contacts, the wedge contacts (jaws) of the top isolator and the busbars.

The incoming cable is connected to the unit busbars, the latter being connected to the contacts of the top isolator. The outgoing cable is connected to the sockets of the "floating"-contact isolator.

On withdrawal of the truck, all the apparatus on it are completely de-energized and may therefore be inspected, serviced or repaired.

An outgoing feeder unit (Fig. 177) houses a circuit breaker provided with overcurrent protection and two meters, one for active
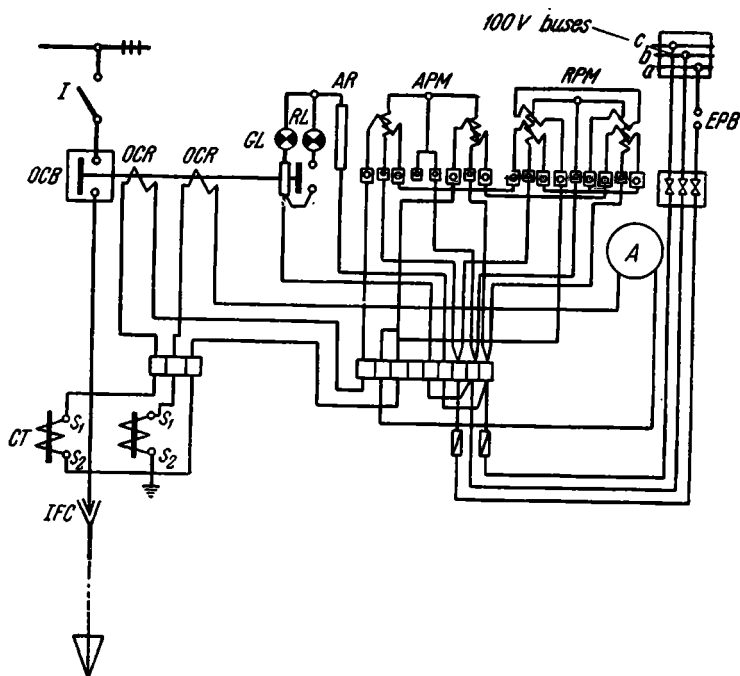
Fig. 177. Circuit diagram of ВЯП-6 (VYaP-6) outgoing-feeder switchgear unit:

I—isolator; *OCB*—oil circuit breaker; *CT*—current transformer; *IFC*—floating contacts of isolator; *OCR*—overcurrent relay; *GL* and *RL*—indicating lamps; *AR*—auxiliary resistor; *A*—ammeter; *APM*—active-power meter; *RPM*—reactive-power meter; *EPB*—emergency push button contacts

power, the other for reactive power. This unit has no built-in voltage transformer; therefore the voltage coils of its meters and the signal lamps receive supply from a 100-volt bus connected to a neighbouring unit having a voltage transformer (for example, an incoming feeder unit).

Units intended for independent use differ from those above in that they are provided with their own voltage transformer and voltmeter. This type of unit is used in mine district substations.

A motor-starting unit does not contain a voltage transformers. At starting, its overcurrent relays are shunted by a special arrangement operated by a foot-pedal linkage movement.
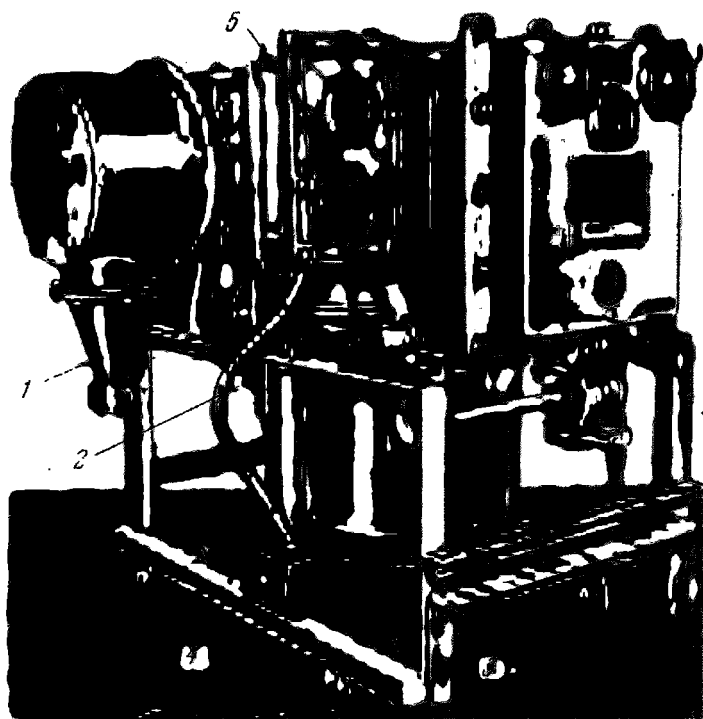
Fig. 178. РВД-6/3 (RVD-6/3) switchgear unit:
1—incoming cable box; *2*—outgoing cable box; *3*—oil circuit breaker; *4*—remote control unit; *5*—meter

When separate units are put together into a switchgear substation their busbars are connected through openings in the side walls fitted with detachable cover plates. The busbars are connected by armoured cable jumpers.

ВЯП-6 (VYaP-6) units have interlocks which prevent the top isolator from being closed or opened, or the truck being withdrawn when the circuit breaker is closed.

Type УРВ-6/3 (URV-6/3) and ВЯП-6 (VYaP-6) switchgear units do not incorporate remote control for their oil circuit breakers (in URV-6/3 units only remote tripping of the supply circuit breaker is possible). The need for remote control of switchgear units has arisen in connection with the introduction of automatic control for

mining electrical installations operated at voltages above 1000 volts, and also with the advent of mobile transformer substations with dry-type power transformers which have to be switched on and off at the supply substation by a control from the mobile substation.

The particular feature of the remote-control scheme adopted in the РВД-6/3 (RVD-6/3) switchgear unit is the use of two solenoids, closing and opening. Both solenoids can be energized from a bank of capacitors charged to 500 V. Pressure on the "close" button energizes a relay which operates to connect the capacitor bank to the closing coil; the latter is energized and thus closes the oil circuit breaker. The circuit breaker is tripped open in a similar manner. Each bank of capacitors is charged through germanium rectifiers from a special step-up transformer. A general view of an РВД-6.3 (RVD-6 3) remote-controlled unit can be seen in Fig. 178.

Rules for Operating Switchgear Rated for Voltages over 1000 V

The oil circuit breakers in the switchgear units of mine district substations are closed and opened by the shift electrician by the order of the district mechanic (in his absence, by the order of the dispatcher or mine electrical engineer).

The shift electrician must have a certificate qualifying him to operate high-voltage equipment. In the event of a fire or accident he has the right to trip circuit breakers without orders and then report to his superiors. In performing the switching of circuit breakers, the electrician must stand on a rubber mat and wear tested dielectric rubber gloves.

If oil is found to leak from a circuit breaker, the latter must not be opened, or an explosion may take place. In such cases the supply to the feeder must be switched off at the central underground substation. After a circuit breaker has opened on a short circuit, it may only be reclosed by the order of the chief mechanic or electrical engineer of the mine.

High-voltage equipment should be visually inspected with power on by the shift electrician at the beginning of each shift. During such an inspection, the load on the feeder units is checked by reading the indicating ammeter; their enclosures, tanks and indicating instruments are examined to see that all is in order; the circuit breaker tank is checked for proper oil level; the cable sealing ends are examined for cable compound leakage; and the earthing connections are checked for proper condition. Also checked for is the presence of the indicating lamps, rubber insulating mats, fire fighting appliances, etc., which must always be on hand in each switchgear chamber. All abnormalities detected during the shift inspection should be entered in the inspection and repair log.

At intervals of not more than one month, a team of electric fitters under the district mechanic shall carry out an inspection and do routine maintenance on receipt of a special permit-to-work.

Prior to the inspection and routine maintenance, it is necessary to open the district substation oil circuit breaker and the feeder circuit breaker in the central underground substation. Then the withdrawal part of the switchgear unit at the central substation is run out to the open position, and warning "Do Not Close—Men at Work" notices are hung on the operating mechanism control levers. Next the withdrawable part of the switchgear unit is drawn out at the mine-district substation, the current-carrying parts must be tested with a high-voltage indicator to see whether they are alive or dead, and a safety earthing conductor set is attached between earth and the incoming-cable terminals in the busbar chamber (if the unit has an angle cable box not filled with cable compound).

The units where the cable boxes are of the compound-filled type, the safety earthing conductor set is not used, but the sockets of the plug isolator must be closed with a metal shutter. Using a safety earthing conductor set, all three phases are shorted to each other and to earth.

In addition to the external inspection, all the contacts must be checked and tightened, the insulators examined for possible damage; the isolators, operating mechanisms, interlocks, circuit contacts looked over for overall condition, and the unit cleaned of dust and dirt. The results of inspection and maintenance must be entered in the equipment inspection log.

Preventive maintenance should be scheduled at three months' intervals, when a team under the supervision of the chief electrical engineer shall check the cable sealing ends and cable boxes, contact connections, insulators, current and voltage transformers, isolators, blade contacts (for possible misalignment, contact pressure, simultaneous closing), breaker interlocks, crossarms and springs. After the inspection is completed all the necessary repairs are made, the separate parts of the units are adjusted, and joint operation of the operating mechanisms and circuit breakers is checked. The quality of the oil and the condition of the insulation shall be also checked.

7-3. Low-voltage Switchgear

In gas or dust hazardous mines, the low-voltage switchgear of inbye substations consists of flameproof feeder air circuit breaker boxes designed to disconnect the low-voltage feeders in the event of a short circuit or a drop in low-voltage circuit insulation resistance.

In the latter case an earth leakage relay is installed in conjunction with the feeder air circuit breaker.

The AФB-1532 (AFV-1532) feeder air circuit breaker box (Fig. 179) is intended for joint installation with an earth leakage relay.

The AФB-1522 (AFV-1522) feeder air circuit breaker box employs the same circuit arrangement and differs only in the size of the contactor used.

Feeder air circuit breaker boxes, types AФB-1521 (AFV-1521) and AФB-1531 (AFV-1531), have no trip coils; they therefore cannot operate in conjunction with an earth leakage relay.

A feeder air circuit breaker box (Fig. 179b) is a three-pole air circuit breaker *CB* mounted on a panelboard and housed in a flameproof enclosure. The supply cable is brought in to terminals L_1, L_2 and L_3 arranged within the cable sealing box. Terminals S_1, S_2 and S_3 connect the outgoing cable to the starter boxes at a distribution centre.

The trip or opening coil is connected across the middle phase and a special terminal *TC* in the terminal box. Interconnection of the circuit breaker with the earth leakage relay is by a flexible cable connected to terminals S_1, S_2, S_3, *TC* and *E*.

The stationary circuit breaker contacts are secured on the insulating panelboard while the moving contacts are assembled on a shaft free to turn in bearings. The moving contact assembly consists of a main, an auxiliary, and an arcing contacts, all connected in parallel. The first to make the circuit on closing is the arcing contact, a component comprising a spring and a renewable copper tip. The auxiliary contact makes next. It consists of a series of tipped copper plates. The last to close is the main contact. The latter is built up in the form of a brush assembled of thin copper laminations. When the circuit breaker is opened, the above contacts open in reverse order.

Moving contacts of such design protect the main contacts from being burned by the arc since the arc is interrupted by the arcing and auxiliary contacts which have tips easy to replace when worn.

The arc is extinguished in these air circuit breakers by deionizing grids. When the arc is drawn on opening it is magnetically blown into an arc chute chamber containing a row of metal plates and is broken up into smaller arcs which cool down and die out quickly. These circuit breakers cannot function without arc chutes because the arc may flash over to a neighbouring phase or to the enclosure.

Such air circuit breakers are fitted with a trip-free mechanism by means of which its contacts are closed and opened. This is a linkage system which permits the breaker to be closed and opened manually.
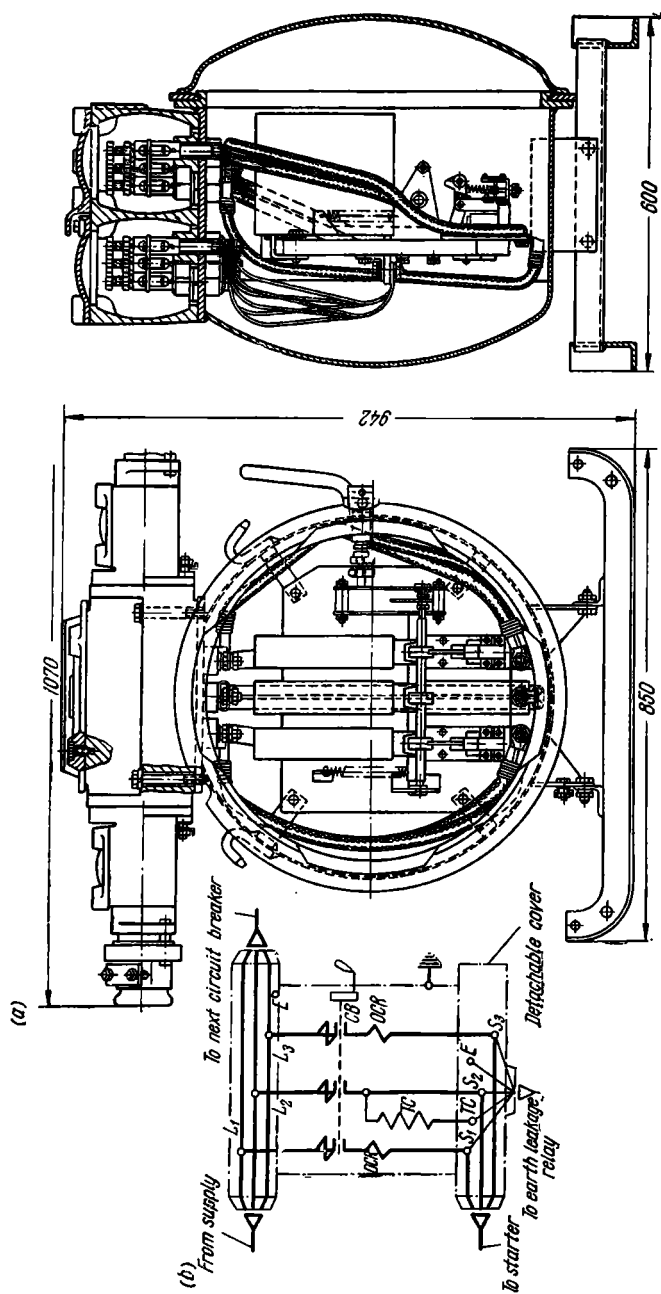
Fig. 179. АФВ-1532 (AFV-1532) feeder air circuit breaker box; *a*—general view; *b*—circuit diagram

For protecting the supply transformer and the outgoing cables from heavy short-circuit currents, the feeder air circuit breaker incorporates instantaneous electromagnetic overcurrent relays in two of the phases to provide automatic trip out.

The overcurrent relay (Fig. 180) comprises a coil *1*, core *2* and a hinged armature *3*. The coil is wound with two turns of copper and is connected in series with the power phase. When a current exceeding the permissible value flows through the coil, the armature is attracted by the core. Its heel comes up against the lever of the trip shaft, the latter turns together with the trip-free mechanism cam, releases the operating lever and trips the circuit breaker open. The value of the current at which tripping occurs (the current setting)* is adjusted by altering the tension of the spring *4* with the aid of a screw *5* and nut *6*. A scale and indicator *7* show the setting of the relay.

The cover in these units is fastened to the enclosure by a bayonet joint, and is so interlocked with the operating handle that it cannot be opened if the circuit breaker is closed.
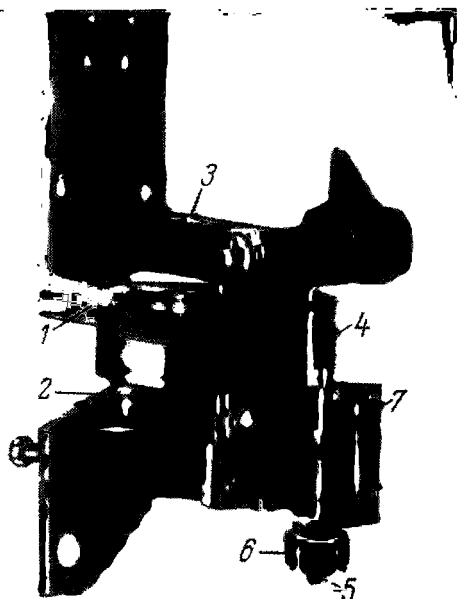
Fig. 180. Overcurrent relay:

1—relay coil; *2*—core; *3*—armature; *4*—spring; *5*—adjusting screw; *6*—adjusting nut; *7*—current scale

The switchgear of a district or inbye substation may in certain cases have to be two or more feeder air circuit breaker boxes. For example, two feeder air circuit breaker boxes will be required for a substation with two transformers or two outgoing feeders.

Type АФВ (AFV) feeder air circuit breaker boxes can be linked together without the use of busbar boxes. This is possible because

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* Current setting is the term used to designate the minimum value of current at which the relay has been set to operate.

they have two supply cable entries, one for the incoming cable from the transformer, and the other for the jumper cable run to the next feeder air circuit breaker box.

Installation and Operation of Feeder Air Circuit Breaker Boxes

The installation of a feeder air circuit breaker box is simply a matter of connecting the necessary cables to it and earthing the enclosure. Where an armoured cable enters the box, the cable sealing end box is compound-filled. In the case of a flexible cable, sealing is accomplished with a rubber packing ring and without compound. If an armoured cable is connected, the lead sheath is connected to the enclosure by means of the clamp provided in the cable fitting and placed over the sheath. The cable armour must be connected to the earthing terminal on the enclosure with the aid of a clamp attached to the cable.

After installation, it is necessary to close and open the circuit breaker several times to make sure that the closing mechanism functions properly. It should be remembered that the current setting of the overcurrent relays must be altered in accordance with changing load or the changing length of the inbye cables.

In external inspection of these feeder circuit breaker boxes, done with the power on, they should be checked for secure clamping of the cables in the cable boxes, possible damage to the cable sheaths and armour, and condition of the interlocks and earthing. The boxes must be not more than 15 degrees from the vertical.

Preventive maintenance should be conducted at intervals not over one week by a maintenance team under the supervision of the district mechanic. This is done with the breakers de-energized. In addition to external inspection, the circuit breaker contacts, trip-free mechanism, relays, arc chutes and contact connections are also examined for possible defects. During this work the burned and fused parts of the contacts are cleaned, all deposits are removed from the arc chute chambers, and all loosened contact joints are drawn tight again.

When setting the arc chutes in place, see that the moving contacts do not catch at their walls or grid plates.

After these boxes have cleared two to three short-circuits on their feeders, they must be given an internal inspection.

It is bad practice to reclose a circuit breaker after a short circuit if the point of fault has not been located. Reclosing of the breaker may result in an open electric arc striking at the fault, capable of causing a gas or dust explosion.

Preventive maintenance should be performed once in one to three months in the mine repair shop.

Selection of Feeder Air Circuit Breaker Boxes and Relay Settings

The selection of feeder air circuit breaker boxes should be governed by the following considerations:

1. The current rating of the circuit breaker should be somewhat higher than the normal working current which is to be carried by the circuit breaker.

2. The overcurrent relay setting must slightly exceed the maximum current likely to be drawn by the circuit to avoid false tripping of the circuit breaker. This maximum current is usually equal to the sum of the starting current of the coal cutter-loader (or coalcutter) motor and the normal working currents of the remaining loads connected to the circuit (conveyors, haulage winches, lighting, etc.).

3. The phase-to-phase short-circuit current* for a fault at the far end of the circuit must be 1.5 times the current setting of the relay.

The magnitude of the phase-to-phase short-circuit current can be determined from the nomogram (or alignment chart) in Fig. 181, in accordance with the power capacity of the district or inbye substation transformer and the length of the cable circuit up to the fault. An example of the use of such a nomogram follows.

Example 17. Select feeder air circuit breaker boxes for the district substation and distribution centre shown in Fig. 182.

The substation supplies the motors of a coalcutter and three scraper-chain conveyors. The rated current of the coalcutter motor is 96A and the starting current is 510A. Each conveyor motor has a rated current of 23A.

1. Normal working current which will flow through the circuit breaker:

$$I_n = 96 + 3 \times 23 = 165A.$$

For this load, we select an АФВ-1522 (AFV-1522) feeder air circuit breaker box for joint operation with an earth leakage relay. This box is designed for a normal current of 200 A, i.e., greater than 165A.

2. Maximum current which can flow in the circuit (occurring when the coalcutter motor is started and the conveyors are in operation):

$$I_{max} = 510 + 3 \times 23 = 579A.$$

The current setting will be $I_{set} = 600A$. It will prevent the circuit breaker from being tripped open when the coalcutter motor is started.

3. Our next step is to check the behaviour of the overcurrent relays in the event of a short circuit at the furthermost point in the circuit (point K_1) from supply.

In the nomogram, the scale on the axis of abscissas gives the length of the cables up to the fault for a cross-sectional area of 50 mm², and also conversion factors for calculating the length of cables with conductors of other cross-sectional area.

---

* A phase-to-phase (or two-phase) short-circuit is a fault which occurs when any two phases (lines) of a three-phase circuit are short-circuited; the two-phase short-circuit current of a circuit is less than the three-phase short-circuit current.

Cable conductor size, mm²	Size conversion factor, K
4	12.30
6	8.22
10	4.92
16	3.06
25	1.97
35	1.41
50	1.00
70	0.72
95	0.54
120	0.43
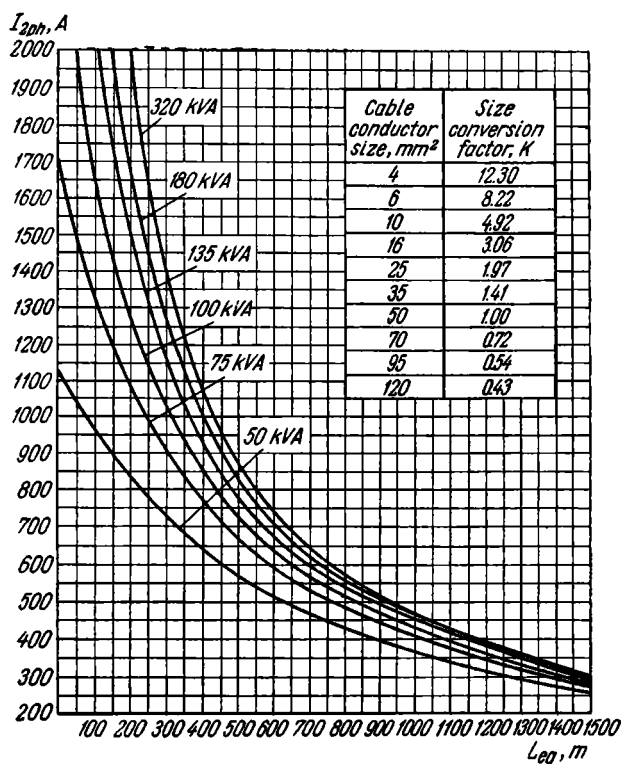
Fig. 181. Chart for determining short-circuit currents
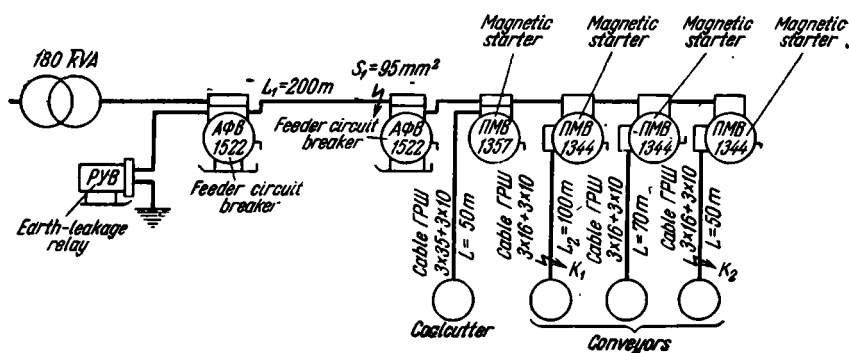
Fig. 182. Diagram for the selection of feeder air circuit breakers

The equivalent length of cables with 95mm² and 16mm² conductors will then be

$$L_{eq} = 200 \times 0.54 + 100 \times 3.06 = 414 \text{ metres.}$$

For this equivalent length, the curve for a 180-kVA transformer will give the two-phase fault current at point K_1, thus:

$$I_{2ph\,(K_1)} = 960A.$$

Since 960A is $960 \div 600 = 1.6$ times greater than the current setting, the relays will reliably trip the circuit breaker, even if the fault occurs at the furthermost point of the circuit.

At any other point in the circuit the short-circuit current will always exceed 960 A and therefore undoubtedly cause the relays to trip the respective circuit breakers. For example, the two-phase short-circuit current at point K_2 for which

$$L_{eq} = 200 \times 0.54 + 50 \times 3.06 = 261 \text{ metres, will be}$$
$$I_{2ph\,(K_2)} = 1375 \text{ A,}$$

a value much greater than the circuit breaker setting (600A).

7-4. The РУВ (RUV) Flameproof Earth Leakage Relay

Earth leakage relays are designed to continuously monitor the condition of the insulation resistance in mining circuits.

The РУВ (RUV) relay operates in conjunction with an АФВ (AFV) feeder air circuit breaker box and will open low-voltage circuits in the following instances:

(1) When a mine worker comes in contact with a live part, and the current which could flow through his body would be able to reach a value dangerous for human life;

(2) When a mine worker comes in contact with a machine enclosure or frame which has accidentally acquired potential when the protective earthing is defective;

(3) When in any one of the circuit phases, damage or failure of the insulation results in a fault to earth.

The following three types of earth leakage relays are available in the U.S.S.R.: РУВ-2 (RUV-2) for 380V circuits, РУВ-2-660 (RUV-2-660) for 660V circuits, and РУВ-2-127 (RUV-2-127) for 127V circuits. As to circuit design and construction, these relays have no essential differences.

These relays receive supply from a selenium rectifier SR (Fig. 183) connected to one of the secondary winding coils (S_0—S_5) of a transformer-choke T-Ch. When the insulation of the circuit is good, the current through the relay R will be insignificant and unable to operate it; the relay contacts R_1 and R_2 therefore remain open. The instant the total insulation resistance of the circuit (i.e., the resistance of all three phases relative to earth) drops to a value of 3500 ohms or less, a direct current sufficient for the relay to operate will flow.
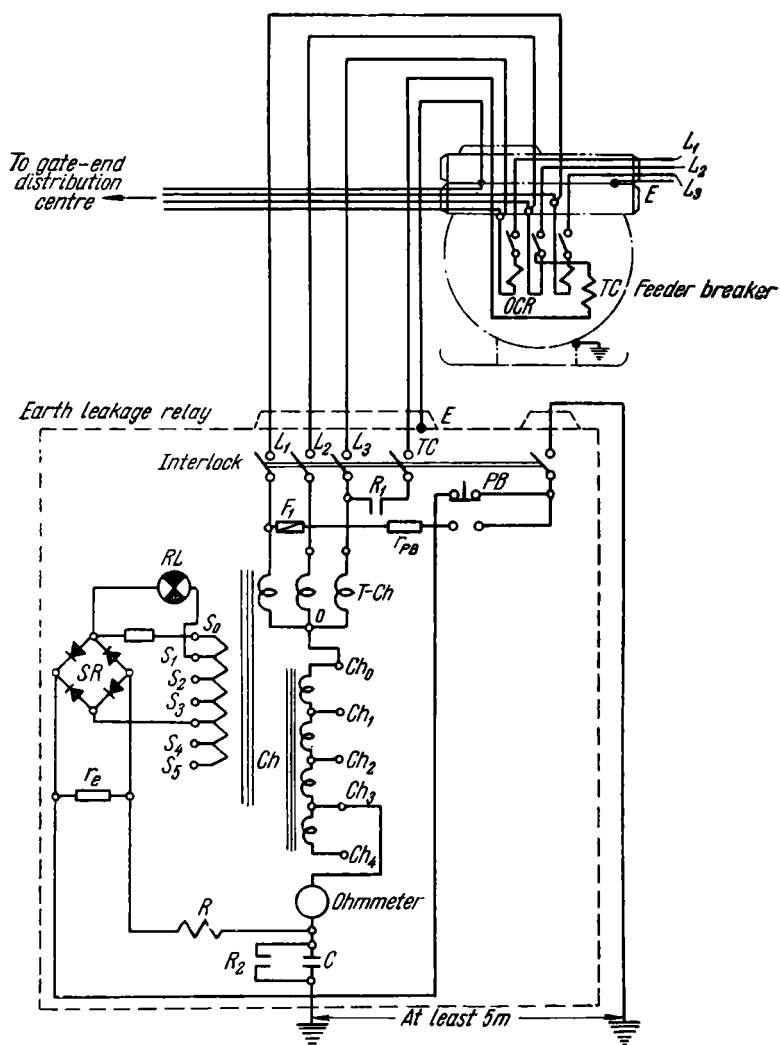
Fig. 183. Elementary circuit of РУВ-2 (RUV-2) earth leakage relay
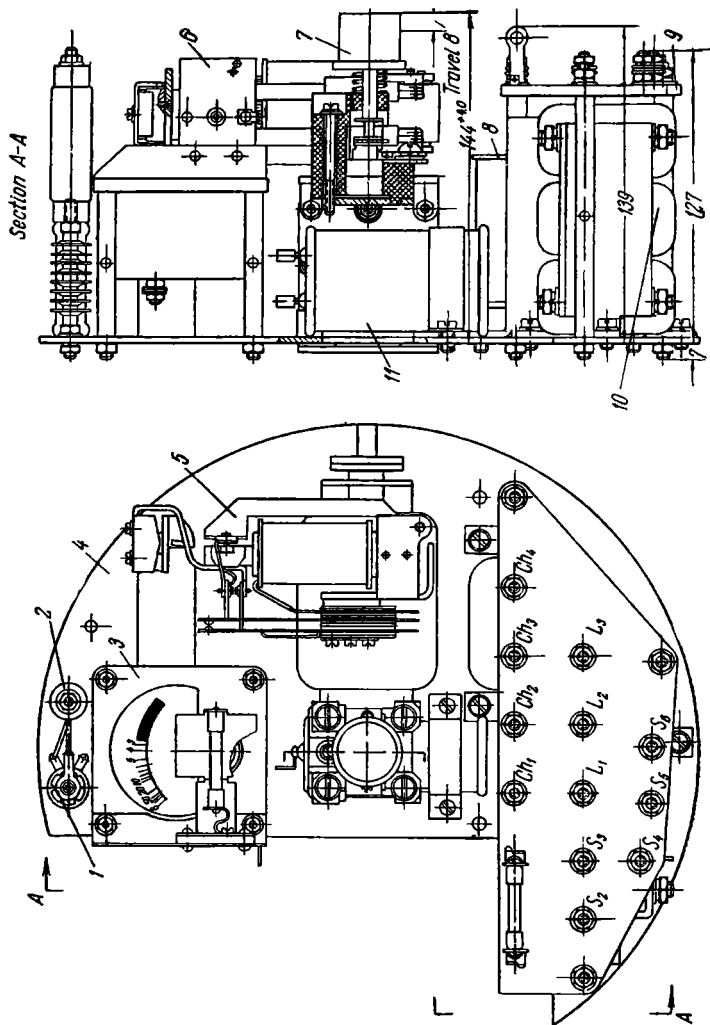
Fig. 184. Withdrawable part of PVB-2 (RUV-2) earth leakage relay:

1—rectifier; *2*—loading resistor; *3*—ohmmeter; *4*—panelboard; *5*—relay; *6*—indicating lamp fitting; *7*—test push button; *8*—choke; *9*—terminal panel; *10*—transformer; *11*—capacitor

The path of this current then becomes: minus terminal of rectifier *SR*, relay winding *R*, ohmmeters, choke winding coils (Ch_4—Ch_0), transformer-choke *T-Ch*, defective phase, earth at the fault, earthing point of the earth-leakage relay, plus terminal of rectifier *SR*. As the relay operates, it closes its R_1 contact in the circuit of the trip (opening) coil *TC* of the circuit breaker to energize the coil and
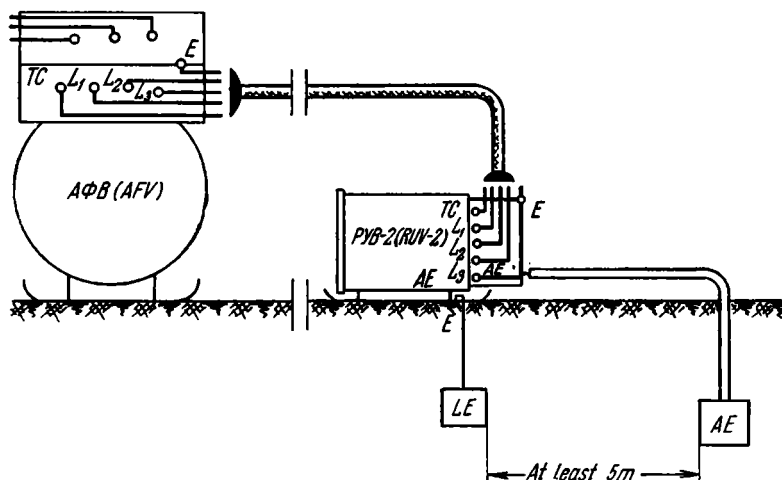
Fig. 185. Connections of earth leakage relay to feeder air circuit breaker

disconnect the faulty circuit. The relay simultaneously closes its R_2 contact. It connects the relay winding directly to the plus terminal of the rectifier by shunting it to earth round the fault in the circuit. Burning of the relay contacts, and erratic operation of the relay is thus avoided when an intermittent or arcing fault occurs.

The function of the transformer-choke *T-Ch* is to interconnect the auxiliary d.c. circuit with the three-phase main or power circuit. An ohmmeter is incorporated to indicate the total insulation resistance of the circuit relative to earth. The push button *PB* provides the means for regularly checking the relay (when *PB* is pressed it faults one of the phases to earth through a resistor r_{PB}).

Fig. 184 shows the removable part of the PYB-2 (RUV-2) relay, while Fig. 185 gives the connections of such a relay with the AФB (AFV) feeder circuit breaker and an auxiliary earthing point *AE*. The earth leakage relay must be installed in the transformer substation and control the main feeder circuit breaker. Its enclosure must be reliably earthed and the auxiliary earthing point must not be closer than 5 metres from the local earthing point *LE*.
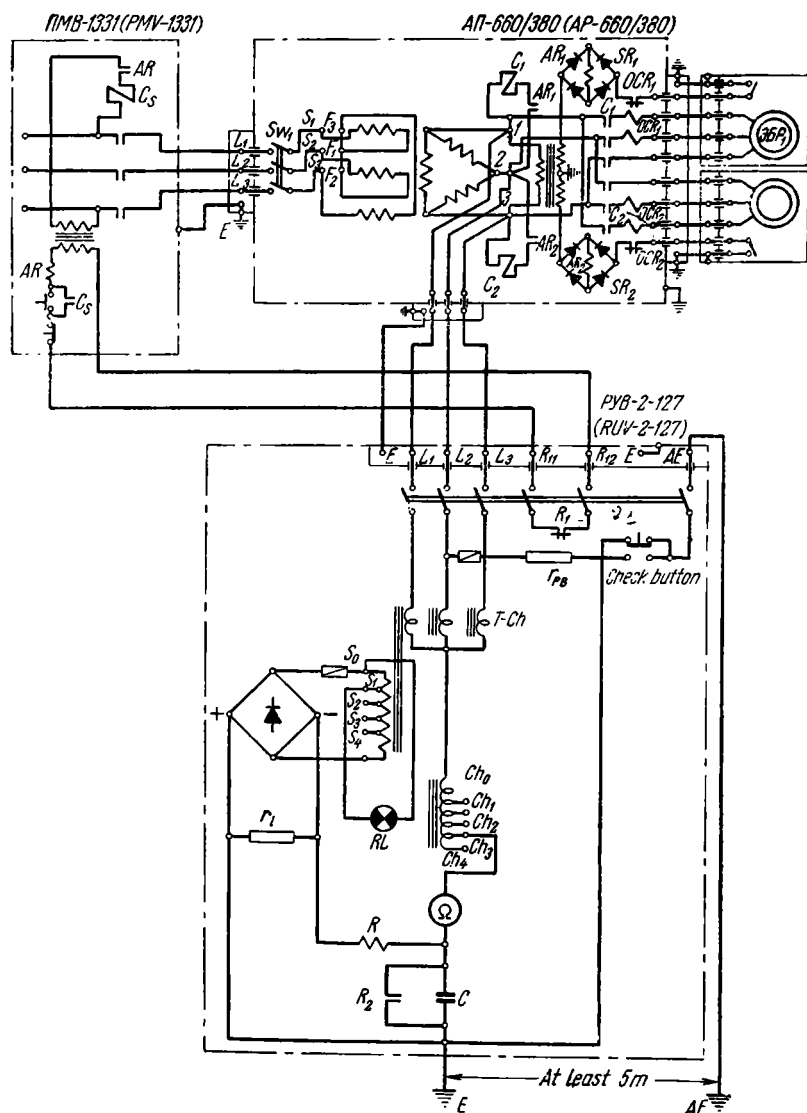
Fig. 186. Connections of РУВ-2-127 (RUV-2-127) earth leakage relay

Whenever the circuit has been opened automatically it is essential to locate the fault. For this, all the manual and magnetic starters at the distribution centre must be de-energized and the feeder circuit breaker re-closed. If the latter trips open again, the fault is evidently in the armoured cable somewhere within the section from the feeder circuit breaker to the distribution centre.

However, if the circuit breaker remains closed, it is necessary to switch on the starters one after another, to determine which of the outgoing circuits causes the circuit breaker to be tripped open. The circuit breaker will also be tripped if, for example, there is a leakage path of 9000 to 10,000 ohms resistance in each of three outgoing flexible cables or motors. Since these leakage paths will be connected in parallel, the total insulation resistance, when the three above cables or motors are connected to the supply, drops to from 3000 to 3300 ohms and causes the earth leakage relay to operate. In such cases, by permission of the mine electrical engineer, the protection may be slightly de-sensitized by lowering the d.c. voltage, accomplished by changing the rectifier supply lead connection to terminal S_4 to S_3 or even S_1.

Fig. 186 gives the circuit connections of the PУB-2-127 (RUV-2-127) earth leakage relay. This relay is designed for use with АП-660/380 (AP-660/380) starter boxes, but can also be used for protecting TCШ (TSSh) lighting transformers.

The line terminals L_1, L_2 and L_3 of the relay are connected to the 127V circuit and its normally-closed contact R_1 is interposed in the auxiliary relay AR circuit of the ПMB-1331 (PMV-1331) magnetic starter through which the starting unit receives its power. On occurrence of an earth fault or an excessive drop in insulation resistance within the 127V circuit, the earth leakage relay (control relay R) immediately operates to open its R_1 contact, de-energize the auxiliary relay AR, and open the magnetic starter.

C. DISTRICT DISTRIBUTION CENTRES

Power from the low-voltage switchgear of the district or inbye substation is transmitted by an armoured cable feeder to the face distribution centre (comprising starters integrated into a single unit) at a gate end.

The distribution centre used to feed the face equipment is moved forward as the face advances every 25 to 50 metres. The distribution centre should also incorporate a feeder air circuit breaker box to have a means for completely de-energizing the centre directly at the gate end.

The separate starter units of a distribution centre are interconnected by short flexible jumper cables (Fig. 187).

As the face or heading is advanced, and its distance from the substation increases, it becomes necessary to increase the length of the armoured cable feeder. To avoid splicing, the cable run
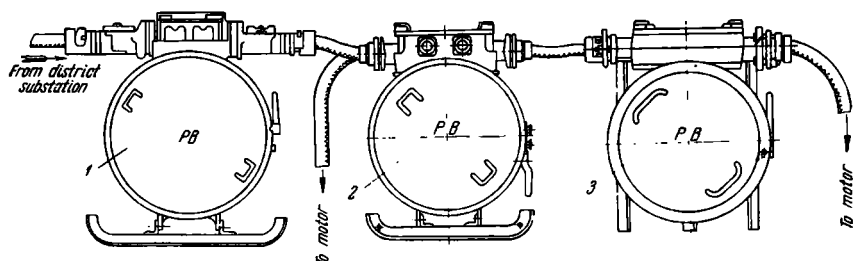
Fig. 187. Flexible connections between gate-end boxes:
1—АФВ-1532 (AFV-1532) circuit breaker box; *2*—ПМВ-1357 (PMV-1357) magnetic starter box; *3*—ПРВ-1007 (PRV-1007) manual starter box

along the gate roadway is taken of such length that it will be sufficient when the distribution centre has eventually reached the farthermost point from the substation. The cable is looped or laid out, as the case may be, and paid out or taken up on each advance or retreat. In paying out and taking up the armoured cable, it must not be bent to a radius less than fifteen outside diameters.

7-5. Gate-end Switchgear

Flameproof Manual Starters

The main types of flameproof manual starters finding application in coal mines are the ПРВ (PRV) and ПБГ (PBG) boxes. They are designed for starting squirrel-cage induction motors with ratings up to 55 kW at 380V. The technical characteristics of these manual starters are summarized in Table 12.

The ПРВ-1007 (PRV-1007) manual starter (Fig. 188) comprises a flameproof enclosure with welded-on cable box, two tubular cartridge fuses *4*, and a hand-operated contactor which has a set of stationary contacts *1* and a set of moving contacts *2* enclosed by arc chutes *3* to aid arc extinction. The contactor is mounted on the rear panelboard. The front panel shielding the contactor carries the cartridge fuses. The starter access cover is fitted to the enclosure body with a bayonet joint and is so interlocked that the cover can be removed only after the starter has been switched off. As the fuses are connected in the circuit after the contactor, they will be dead when the cover is removed and may be replaced with safety. The parts of the starter still remaining alive are shut off by the front panel.
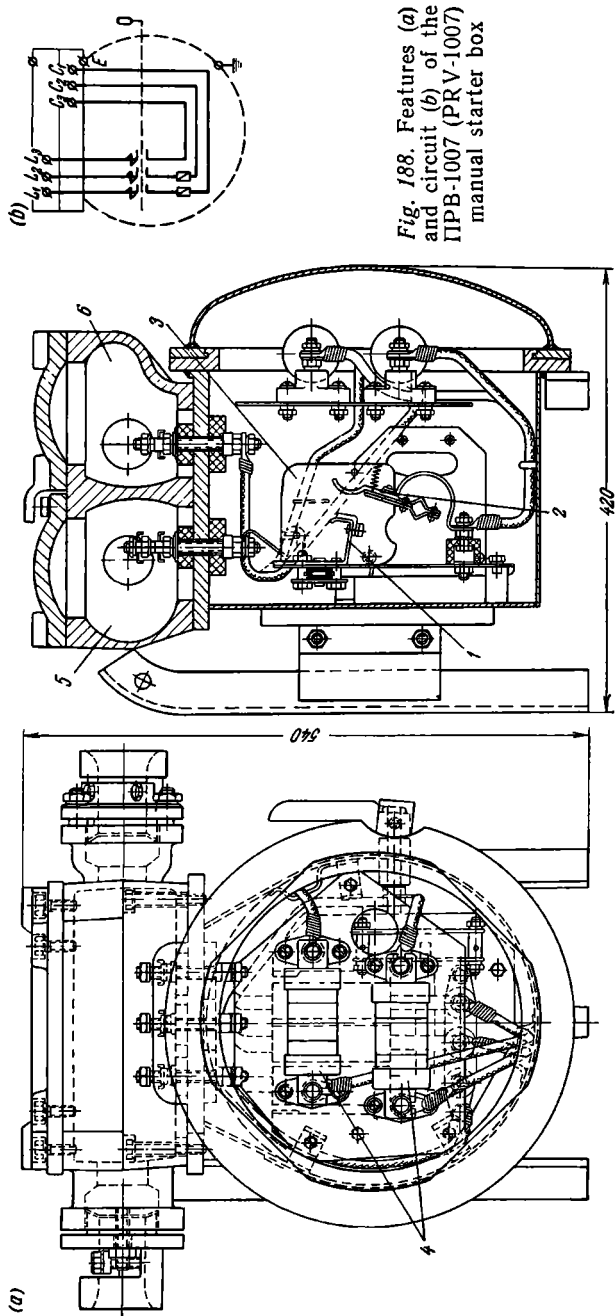
Fig. 188. Features (*a*) and circuit (*b*) of the ПРВ-1007 (PRV-1007) manual starter box

11*

Table 12

Starter	Rated current, A	Maximum kW rating of motor		Built-in protection	Cable connections	Grade and largest size (mm²) of cable	Weight, kg
		220 V	380 V				
ПБГ-380 (PBG-380)	15	4	4	Fuses	Single-way plug socket	ГРШ 3×16+1×10	31
ПРВ-1007 (PRV-1007)	80	30	40	Fuses	Fixed cable box	ГРШ 3×35+1×10	75
ПРВ-1031 (PRV-1031)	60	15	25	Fuses	as above	as above	60
ПРВД-1013 (PRVD-1013)	100	32	55	Fuses	as above	ГРШ 3×50+1×10	61

The starter has a double-chamber cable box. One (entry) chamber *5* has a cover marked "Supply" and the other *6* is for connection of the motor cable. The "Supply" chamber has two openings so that starter boxes can be interconnected without busbar boxes. Attached to the cable box, are sealing-ends for armoured or flexible cables. They are fitted with rubber packing rings and a flexible plate. For convenience of movement, these starters are mounted on skids.

The ПРВ-1031 (PRV-1031) manual starter differs but slightly from the ПРВ-1007 (PRV-1007) in design.

In the ПРВД-1013 (PRVD-1013) manual starter (Fig. 189) power from the mains terminals *1* is applied to fuses *2* through plug and socket isolators *3*. Therefore, when the cover carrying the fuses is swung out, all parts of the starter are de-energized. To the terminals *4* (to which the motor cable is connected) the voltage is applied through the contactor *5*.

The ПБГ-380 (PBG-380) starter (Fig. 190) is designed for manual control of nonreversing motors driving low-power equipment (post-mounted electric drills, loaders, etc.).

A four-core cable applies the supply voltage to the terminals L_1, L_2 and L_3 located in the entry box of the starter, and the earthing core is connected to the terminal E. The three-phase drum switch Sw and fuses F bring the supply up to the plug socket PS. The outgoing cable is connected to the power terminals C_1, C_2, C_3 in the plug; its earthing core is connected to the terminal E. The terminal E_2 serves for earthing the starter enclosure.
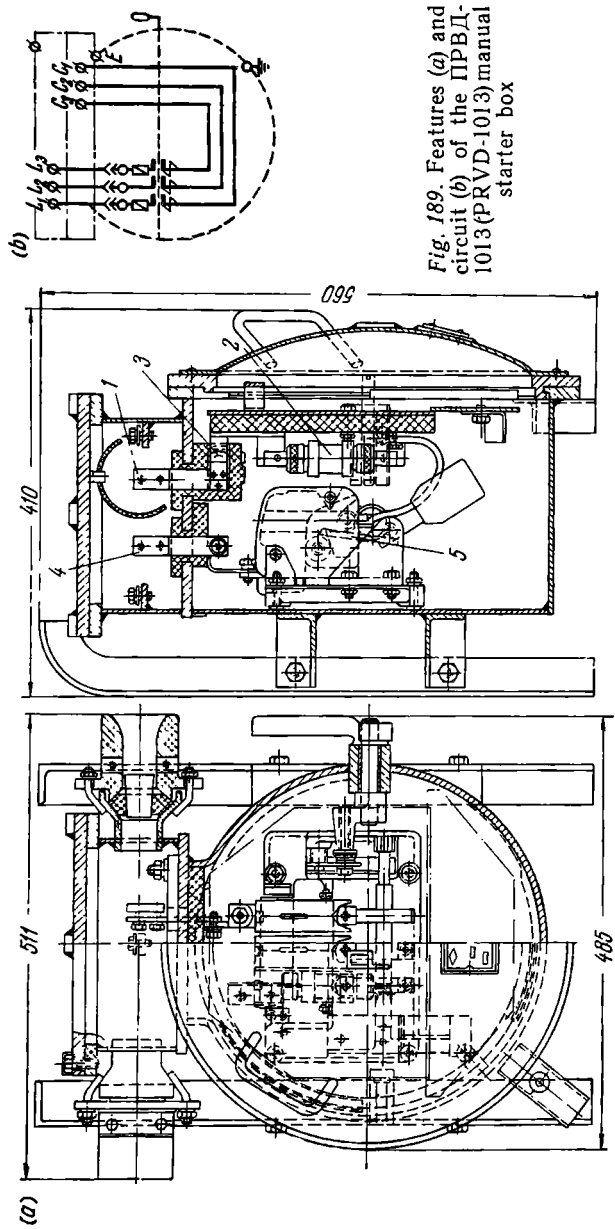
Fig. 189. Features (a) and circuit (b) of the ПРВД-1013 (PRVD-1013) manual starter box
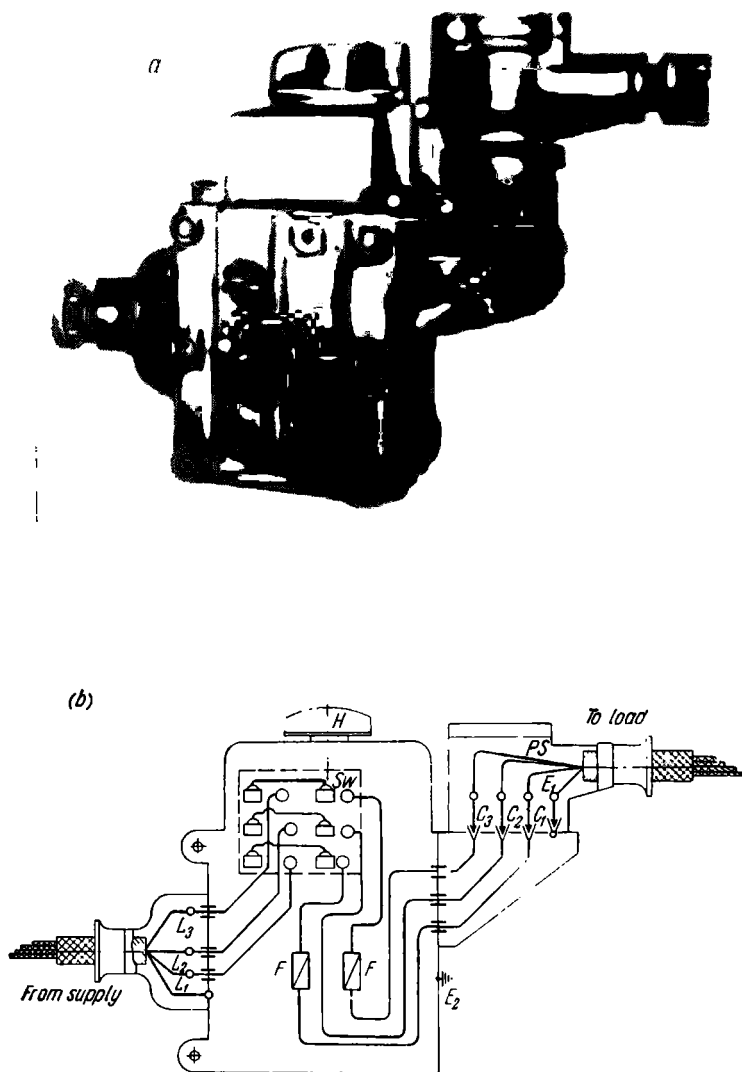
Fig. 190. General view (*a*) and circuit (*b*) of ПБГ-380 (PBG-380) manual starter box

Magnetic Starters for Remote Control of Coal Mining Equipment

By the remote control of coal mining equipment is meant the remote starting and stopping of drive motors by means of magnetic starters. Such starters are remotely controlled from push button stations or by means of special interlock contacts fitted in controllers.

Remote control in mines with gas and coal-dust laden atmospheres uses flameproof magnetic starters, types ПМВ-1331 (PMV-1331), ПМВ-1344 (PMV-1344), ПМВ-1357А (PMV-1357A), ПМВ-1365 (PMV-1365) and reversing magnetic starters, types ПМВР-1441 (PMVR-1441), ПМВР-1451 (PMVR-1451).

A magnetic starter box is built around a three-pole contactor which operates in conjunction with several other devices (such as fuses, relays, control push buttons, etc.), and all are mounted on panels within a flameproof enclosure. The contactor is closed by an operating coil. The enclosure cover is fitted to the body with a bayonet joint. On the right-hand side of the enclosure will be found an isolating switch handle and the "Start" and "Stop" push buttons, so that the starter can be switched on and off directly at the enclosure. The enclosure cover and the isolating switch handle are so interlocked that the cover cannot be removed if the isolating switch has not been opened. After the latter has been opened, none of the parts on the front side of the panel remain alive.

The cables are led in and out of the enclosure through cable boxes carrying sealing-end fittings for clamping and sealing the cables. Starter boxes fitted with solid sealing-end fittings can only take flexible cables; those fitted with combination sealing-end fittings are suitable for connection of both flexible and armoured cables. The control circuit in these starters is fed at 36 volts through a step-down transformer. This transformer is mounted on the back wall of the panel.

All the magnetic starter boxes, as to control circuit arrangements, may be divided into two groups: those employing no intermediate or auxiliary control relay, and those incorporating such a relay.

In magnetic starter boxes having no auxiliary control relay [ПМВ-1344 (PMV-1344), ПМВ-1441 (PMV-1441)] the operating coil of the contactor is energized by a step-down (36 V) transformer. Since the current drawn by the operating coil is relatively large on a closing, a considerable voltage drop occurs in the control circuits. This naturally limits the length of the control circuit with which reliable starter control can be obtained.

In magnetic starter boxes which incorporate an auxiliary control relay, the operating coil is supplied at the power circuit voltage (380 V) and is energized by closing of the contacts of the auxiliary relay whose coil is energized first by supply from a step-down trans-

former. The pick-up current of the auxiliary relay is very much lower than that of the operating coil. Starters incorporating an auxiliary relay therefore operate with full reliability even on long control circuits.

The cable led out of a magnetic starter box to a motor is protected from short circuits either by fuses or by overcurrent relays [ПМВ-1365 (PMV-1365), ПМВ-1365A (PMV-1365A)]. Both the fuses and the overcurrent relays are inserted in two of the phases. Both relays in a box have their normally-closed contacts connected in series with the operating coil of the contactor. When either of the relays operates, its contacts open to break the coil circuit, and the contactor opens.

In examining the circuits of magnetic starter boxes, we will make use of elementary or so-called "developed" circuit diagrams which are a great aid in the study of the operation of any circuit.

The ПМВ-1344 (PMV-1344) magnetic starter box is designed for the remote control mainly of conveyor motors, and also of coal-loading machine, winch and other motors. This starter box (Fig. 191) is housed in a flameproof enclosure provided with incoming and outgoing cable entries. The

Fig. 191. General view of ПМВ-1344 (PMV-1344) magnetic starter box

upper cable box is generally fitted with two sealing-end fittings, one for leading in the power supply cable, and the other for connection to the next magnetic starter integrated with it at a distribution centre. The side cable box has a sealing-end for the cable led out to the motor. For connection of the control circuit cables, both cable boxes have cable glands. To make the starter convenient to move from place to place, it is supplied with skids.

The closing and opening of the power circuit is effected by contactor K (Fig. 192), closed by operating coil K fed at 36 V from the secondary winding of step-down transformer Tr. The dotted lines in the elementary diagram show how the "Start" and "Stop" push buttons are connected. In addition to the power or main contacts, the contactor shaft also carries two normally-open interlock contacts K_1 and K_2. When the contactor is closed, they are also closed.

Elementary diagram of starter connections

Fig. 192. Internal connections of ПМВ-1344 (PMV-1344) magnetic starter box

Within the incoming cable box (Fig. 193) can be seen three terminals L_1, L_2 and L_3 for connection of the supply cable; terminals *9* and *13* for connection of the control cable (when the starter must be interlocked with its neighbours); and terminal *E* for connecting the cable earthing core.

The side cable box (Fig. 194) contains three terminals C_1, C_2 and C_3 to which the motor cable cores are connected; terminals *1, 2* and *8* for connecting a control push button station cable or a
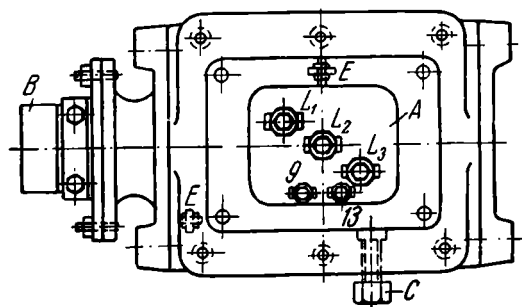
Fig. 193. Upper cable box of ПМВ-1344 (PMV-1344) magnetic starter box:

A—intermediate box; *B*—trailing cable gland; *C*—gland for flexible control cable
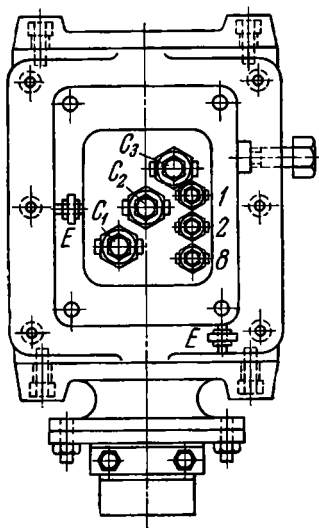
Fig. 194. Side cable box of ПМВ-1344 (PMV-1344) magnetic starter

controller interlock circuit cable, and terminal *E* (earth) for connection
to the motor frame. If the starter is used to control a single motor,
the following jumpers must be put in (see Fig. 192): one between
terminals *9* and *E* in the top cable box; one between terminals *2*
and *5* on the panel (jumpers J_c and J_b, respectively), and a jumper
J_a for connection of a lighting unit.
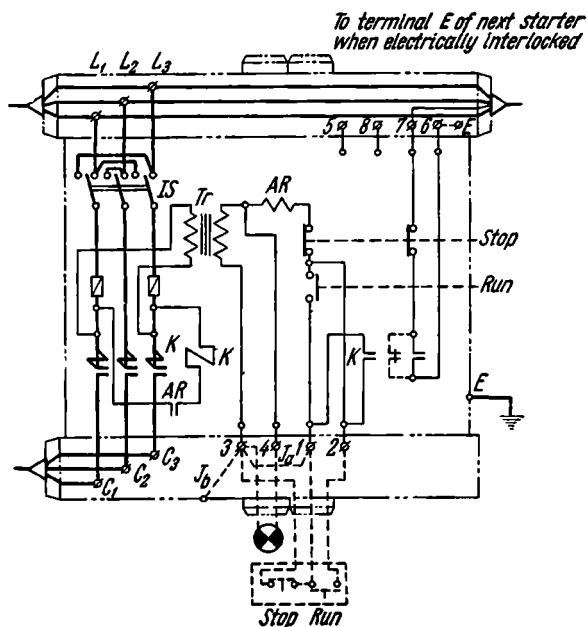
Fig. 195. Internal connections of the ПМВ-1331
(PMV-1331) magnetic starter box:

A — jumper used when remote control box unit is not used;
B — jumper used for earthing

Pressure on the external "Start" button energizes the operating
coil of the contactor through: the following current path terminal *4*
of transformer *Tr*, starter (internal) "Stop" button, contactor coil *K*
(terminal *1*), control cable core, external "Start" button, external
"Stop" button, control cable core, starter earth terminal *E*, terminal
9, jumper and transformer *Tr*. Contactor *K* is thus closed and ap-
plies the supply voltage to terminals C_1, C_2 and C_3; interlock con-
tacts K_1 and K_2 also close. The "Start" button can now be released,
as coil *K* is fed through interlock contact K_1 connected in parallel
with the "Start" button. Pressure on the "Stop" button will open the
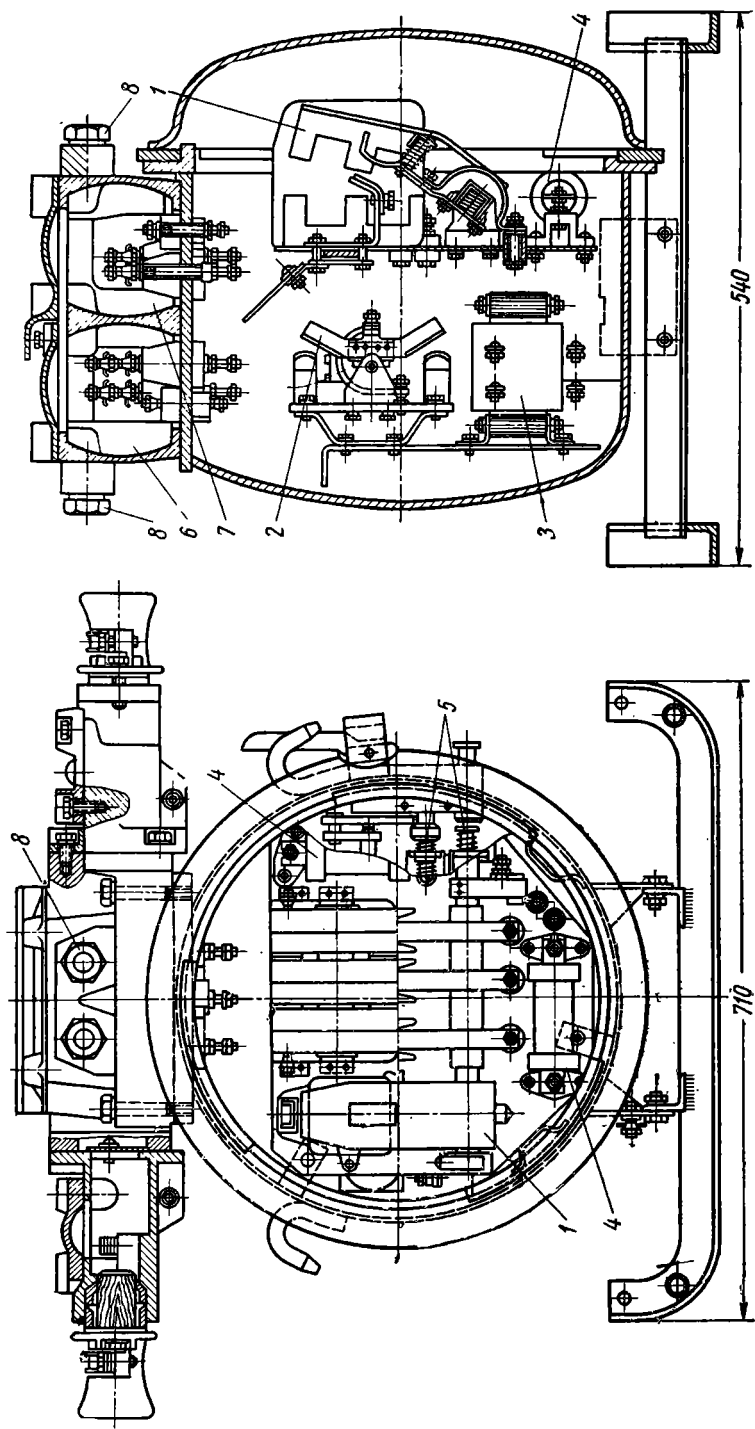
Fig. 196. Parts of ПМВ-1357A (PMV-1357A) magnetic starter box:

1—contactor; *2*—isolator; *3*—transformer; *4*—fuse; *5*—push button; *6*—incoming cable box; *7*—outgoing cable box; *8*—glands for control cable

starter. Terminal *13* is used when the starter has to be interlocked with other starters.

The ПМВ-1331 (PMV-1331) magnetic starter box is intended for remote control of conveyor motors. It has an auxiliary relay. When the external "Start" button is pushed (Fig. 195), a current will be passed through auxiliary relay *AR* around the following path: transformer *Tr*, winding of auxiliary relay *AR*, starter (internal) "Stop" button, terminal *2*, "Start" and "Stop" buttons of the push button station, terminal *3* and transformer *Tr*. Auxiliary relay *AR* immediately picks up to close its contact in the circuit of contactor operating coil *K*, and energizes it to close the contactor. As soon as the contactor closes, its interlock *K* shunts the "Start" button. The local lighting lamp connected to the starter box receives its supply from the step-down transformer.

The ПМВ-1357A (PMV-1357A) magnetic starter is intended for remote control of the motors of coalcutters, cutter-loaders, and also of winches and conveyors. Fig. 196 gives a general view, and Fig. 197, the internal circuit connections of this starter box.

The contactor coil is supplied at the power circuit voltage, but the control circuit operates at 36V, provided by a 380/36V step-down transformer. This starter incorporates an auxiliary relay. The incoming and outgoing cable boxes are arranged side by side in one plane at the top of the enclosure. Also provided is a double-throw
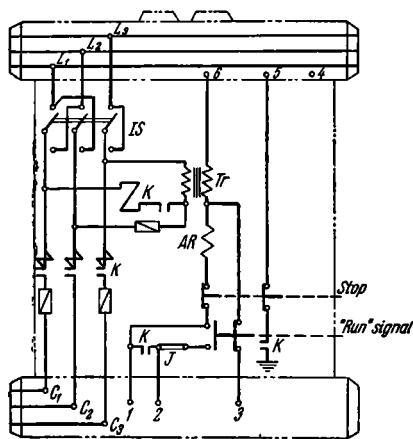
Fig. 197. Internal connections of ПМВ-1357A (PMV-1357A) magnetic starter box:

IS—isolating switch; *K*—contactor; *Tr*—step-down transformer; *AR*—auxiliary relay; *J*—strap
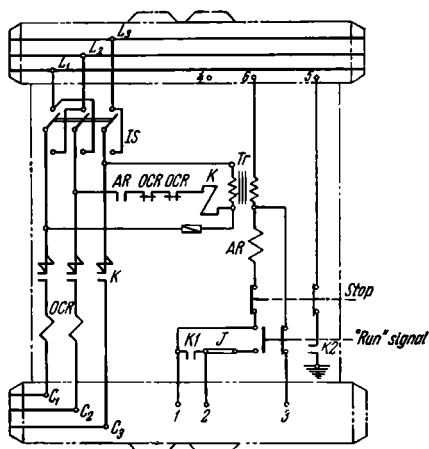
Fig. 198. Internal connections of ПМВ-1365 (PMV-1365) magnetic starter box

isolating switch for reversal of the connected motor and for disconnecting the apparatus on the starter panel from the supply.

ПМВ-1365 (PMV-1365) magnetic starter boxes serve to remotely control the motors of powerful coalcutters, cutter-loaders and also winch and pump motors.

Their incoming and outgoing cable boxes are set side by side in one plane at the top of the enclosure. Combination sealing-end fittings are fitted to permit connection of either flexible or armoured cables. In the latter case, the sealing end is compound-filled. The support panel in this starter is of metal and the apparatus is insulated from it by mica-bakelite spacer bars coated with an oil-resistant enamel.

Contactor coil K (Fig. 198) is fed directly from the power circuit and is connected through the normally-closed contacts of overcurrent relays OCR inserted in two of the outgoing phases, and through normally open auxiliary contact AR of the auxiliary relay.

The auxiliary relay is fed from the secondary of step-down transformer Tr. The function of interlock K_1 is to shunt the "Start" button, and that of interlock K_2 to interlock the given motor starter with the associated motor starters. To protect the step-down transformer and contactor operating coil, a 6A fuse is included.

The ПМВР-1441 (PMVR-1441) reversing magnetic starter box is intended for the control of reversing mechanisms (flitting and car-spotting winches, incline haulage winches, etc.).
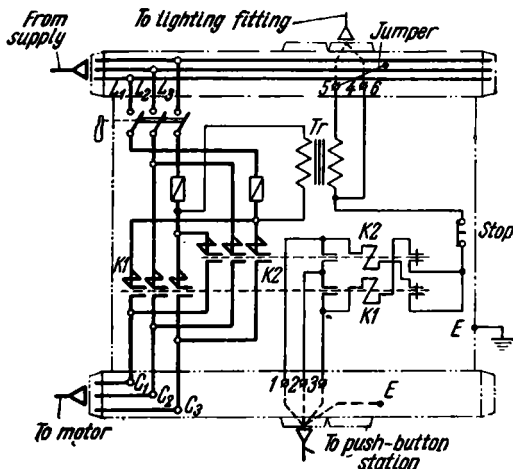
Fig. 199. Internal connections of ПМВР-1441 (PMVR-1441)magnetic starter box

This box (Fig. 199) contains two three-pole contactors $K1$ and $K2$ for "forward" and "reverse" control. They are interlocked both mechanically and electrically so that closing of any one of them prevents closing of the other. This precaution has been necessitated by the fact that simultaneous closing of both contactors would lead to a "short" in the power circuit within the starter. Electrical interlocking is provided by normally-closed interlock contacts. If one of the contactors is closed after its operating coil is energized (coil $K1$, for example), its normally-closed interlock $K1$ inserted in the circuit of the second contactor operating coil $K2$ will open to preclude energizing of the coil.

Therefore, to close contactor $K2$, it is first necessary to open contactor $K1$. When this is done, the interlock of the latter recloses to complete the supply circuit to contactor coil $K2$. In this starter box, a step-down transformer Tr is used to energize the operating coils. For connection of a local lighting unit and signalling devices, a set of terminals is also included. The isolating switch is of the non-reversing three-pole type. Both starter-enclosure access covers (front and rear) are interlocked with the isolating switch handle to prevent their removal when the switch is closed. Only a single "Stop" button is provided. Starter control is accomplished with a three-button КУВ-6013 (KUV-6013) push button station ("Forward", "Reverse", "Stop"). To permit connection of either flexible or armoured cables the incoming and outgoing cable boxes are furnished with combination type sealing-end fittings.

ПМВР-1451 (PMVR-1451) reversing magnetic starter boxes provide a means for remote motor starting, stopping and reversal.

This box is of flameproof design with intrinsically-safe control circuits. The apparatus is de-energized by means of an isolating switch. Motor starting, stopping and reversal is accomplished by the "Forward" and "Reverse" line contactors. The starter enclosure is fitted with two interchangeable access covers mechanically interlocked with the isolating switch handle and the "Stop" button to prevent removal when the switch is closed.

The combination type sealing-end fittings provided on the starter permit connection of either flexible or armoured cables.

Line contactors $K1$ and $K2$ (Fig. 200) are closed by means of d.c. auxiliary relays AR_1 and AR_2 fed from selenium rectifiers SR. The latter operate on 36 V supplied by the secondary of a step-down transformer Tr. Control of these starters is accomplished with a КУВ-6013 (KUV-6013) three-button station.

The starter circuit functions in the following way: with isolating switch IS closed and the "Forward" button depressed, the following current path is completed: terminal a of transformer Tr, fuse F_3, starter "Stop" button, terminal b of selenium rectifier SR_1, terminal

c of the same rectifier, normally-closed interlock *K*2 of "Reverse" contactor (the interlock *K*2 remains closed because the "Reverse" contactor has not been closed), normally-closed upper contacts of the "Reverse" push button, closed lower contacts of the "Forward" push button, normally-closed contacts of control-station "Stop" button, earth terminal *E* in the outgoing cable box, starter enclosure, earth terminal *E* in the incoming cable box, jumper *E*-6, and the opposite side of the transformer secondary.

The establishment of the above path energizes auxiliary relay AR_1 with direct current, the relay picks up and closes its contact in the circuit to the operating coil of the *K*1 line contactor; the main contacts *K*1 close to apply the supply voltage to the motor. Contactor interlock *K*1 closes simultaneously and thus allows the "Forward" button to be released because it maintains the coil current flow through itself. If the motor must be reversed, the "Stop" button must be pushed to stop the motor and the "Reverse" button depressed.

If the "Reverse" button is pressed before the "Stop" button is (i.e., with the mo-

Fig. 200. Internal connections of ПМВР-1451 (PMVR-1451) magnetic starter box: *IS*—isolating switch; F_1, F_2 and F_3—fuses; *Tr*—transformer; *SR*—selenium rectifiers; *K*1 and *K*2—"Forward" and "Reverse" line contactors; AR_1 and AR_2—auxiliary relays and their contacts

tor still running under power) the "Reverse" contactor *K*2 will not be able to close because the selenium rectifier SR_2 is disconnected from the transformer secondary by the open interlock *K*1. Such electrical, interlocking guards against a power short circuit which would result from a simultaneous closure of both contactors.

Should an attempt be made to start the motor by simultaneously pushing both the "Forward" and "Reverse" buttons, the starter will not operate, since the supply to selenium rectifiers SR_1 and SR_2 is cut off by the open upper contacts in both push buttons.

In addition to the electrical interlocks, both contactors are also mechanically interlocked to prevent simultaneous closing. Power and control cable protection is provided by fuses.

Two fuses F_1 protect the flexible motor cable from short circuits; fuse F_2 protects the supply control circuit from short circuits in the contactor coils and in the step-down transformer primary, and fuse F_3 protects the step-down transformer secondary against any short circuits which may develop in the associated circuits.

Owing to the intrinsical safety of the remote control circuit of this starter it is possible to use it in gas and dust hazardous mines for setting up remote control systems which continuously monitor the continuity of the cable earth conductor without danger of sparking. Safety from spark hazards is ensured by low working and short-circuit currents in the control circuits and also by the fact that dangerous emfs of self-induction cannot be induced in the auxiliary relay windings, since they are energized by selenium rectifiers.

7-6. Maintenance of Magnetic Starter Boxes

Inspection of a Box Prior to Sending it Underground

Every magnetic starter box, before it may be taken underground, must have the following operations performed on it.

1. The interlock between the box enclosure cover (covers) and the isolating switch handle must be checked for proper functioning. To do this, the starter "Stop" button is held down and the switch handle is simultaneously thrown into its middle position. The interlocking screw is then screwed in until it goes clear of the slot in the enclosure cover. This permits the two cover handles to be turned counterclockwise to free the covers and then take them off.

2. The arc chutes should be taken off, the panel wiped clean and the coat of vaseline removed from the core of the operating coil. The mating faces of the armature and fixed core can then be checked for intimate fit over the entire area of their faces by pressing the moving assembly by hand against the fixed parts. The power and the interlock contact surfaces are also checked for adequate contact in the same way. The arc chutes are then put in place and the operation of the contacts is checked to see that the chutes do not interfere with the moving system.

3. The fuse links are inspected for correct current rating of their fusible elements.

4. The insulation resistance of the power circuit and the control circuit should next be measured with a megohmmeter (the respective minimum permissible values are 400 and 40 kilohms).

The insulation resistance of the main circuit is measured with the megohmmeter connected, in turn, between each of the input and output terminals (L_1, L_2, L_3 and C_1, C_2, C_3) and the respective earth terminal in the cable box.

The insulation resistance of the control circuits should be measured between each terminal in the cable box and the earth terminal E. Boxes with a low insulation resistance must be handed over for repair.

5. All the jumpers must be checked for connection across proper terminals.

After all the above operations have been performed, the enclosure is closed with its cover and the box is checked by trial operation. This is done by applying a voltage to terminals L_1, L_2 and L_3, closing the isolating switch, and performing several closing and opening operations by means of the push buttons on the box.

Installation of Magnetic Starter Boxes in Coal Mines

1. ПМВ (PMV) magnetic starter boxes operate reliably when their angle of inclination from the vertical is not over 15 degrees. The floor on which they are placed should therefore be as level as possible. They should be placed in a location protected from dripping moisture.

2. The box must be arranged so that the isolating switch handle will be easy to operate and room is provided for removal of the enclosure cover.

3. Box enclosures must be connected to a local earthing. Furthermore, the starter enclosure must be connected to the general earthing system through the lead sheath of the armoured cable or the earthing conductor in the flexible cable used to feed the starter.

4. Armoured cables must be terminated at starter boxes so that their conductors are not unduly tensioned. The compound-filling neck on the sealing-end fitting of an armoured cable box must always be closed by a cover.

An armoured cable is terminated in a sealing-end fitting (Fig. 201) as follows. The armour, coverings and lead sheath are removed from the end of the cable for a length sufficient for sealing and making the connection; each conductor is served, over its insulation with insulating tape up to the end; each conductor is slipped through the respective hole in the insulating spacer plate; the clamp with which the cable is secured in the fitting is put on over the armoured end of the cable and a strap clamp is fitted to press the lead sheath to the box body so as to provide a continuous earth circuit; the cable armour is connected by a clamp and an earthing jumper to the earthing bolt provided on the external surface of the fitting.

For reliable insulation of the conductors, the sealing-end fitting is filled with cable compound through the front opening, and the fitting is closed by its cover.*

A flexible cable is terminated in the sealing-end fitting (Fig. 202) by going through the following steps. The tough outer sheath is
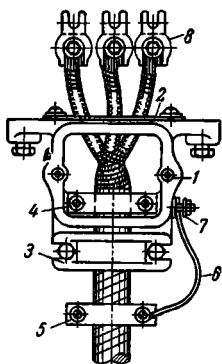
Fig. 201. Termination of an armoured cable in a sealing-end fitting:

1—sealing-end body; *2*—insulating plate; *3*—clamp for securing cable over armour; *4*—clamp for earthing lead sheath; *5*—clamp for earthing cable armour; *6*—jumper conductor; *7*—earthing screw; *8*—cable connector
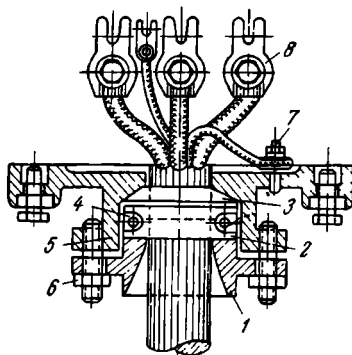
Fig. 202. Termination of a trailing cable in a sealing-end box:

1—belled-throat compressing collar; *2*—clamping half-ring; *3*—rubber packing ring; *4*— screws for joining and tightening half-rings; *5*—stuffing box recess; *6*—nuts for securing compressing collar; *7*—ear hing screw; *8*—cable connector

split for a length sufficient for the connection; the split portion is trimmed off and the belled-throat compressing collar, clamping half-rings and a rubber packing ring are slipped over the cable and the clamping half-rings tightened by screws. The cable end is next inserted in the sealing-end fitting so that the rubber packing ring and the half-ring clamp enter the packing box, and the compressing collar is slid up into place and pulled tight with nuts. The earthing conductor of the cable must be attached to the screw provided on the inside surface of the fitting body.

Before any cable conductor is connected to a terminal it should be checked to make sure that it is at the right terminal. An improper connection can lead to an emergency.

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* A new method is to terminate and seal the cable without a filling compound. By this method, the sealing is accomplished with glass fabric materials and a special silicone varnish.

5. When the installation work on a box has been completed, all the access covers must be replaced (on the box, motor, and push button stations). Only after this may the voltage be applied. Under no circumstances shall a cover on energized apparatus be opened, or the connections of a live cable conductor changed.

6. Motors must be checked before operation for proper direction of rotation. With boxes fitted with a reversing isolator switch reversal of the motor is effected by simply throwing the switch handle over to its opposite position while holding the starter stop button down. For motor reversal where the box is fitted with a single-throw (on-off) isolating switch, the isolating switch must be opened and any two of the phase leads interchanged either in the starter cable box, or in the motor terminal box. If the motor is located at some distance from the starter box, the switch must be opened, a "Do Not Switch On!" notice must be hung on its handle and someone stationed to keep watch on the box. Only then may the motor leads be reconnected.

Inspection and Preventive Maintenance of Magnetic Starter Boxes

Magnetic starter boxes should be inspected fully de-energized at intervals of not longer than one month. Only where it is impossible to completely de-energize the box is it permissible to inspect and make minor repairs on the face side of the panel with the isolating switch open. It must be remembered that the terminals in the incoming cable box remain alive in such instances.

During the inspection the contacts must be looked over to see that they are clean. When beads of fused metal or pitting are detected on them, they should be smoothed down with a fine-cut file.

The power or main contacts in magnetic starters must not be greased. The grease will burn in the arc and deposit carbon on the contact surfaces. The surfaces of interlock contacts are cleaned by wiping them with a cloth. The arc chutes must be kept securely held in place and fixed so that they do not catch at the moving contacts. Starters must never be operated without arc chutes, as this will result in severe burning of the power contacts and can bring about a phase-to-phase arc-over.

Special care must be taken to maintain the closest possible fit between the mating faces of the armature and the operating-coil core. Even the smallest air gap will cause the coil to draw an excessive current and quickly burn out.

The shading coil (the short-circuited ring on the magnet core) must be inspected to see that it is in place and securely seated.

Magnetic Starter Box Troubles, Their Causes and Remedies

Trouble	Cause	Remedy
Starter fails to close when "Start" button is pressed	Blown fuse Open isolating switch Open control circuit "Start" button contacts fail to close; push rod jams	Replace fusible element Close isolating switch Locate and eliminate open-circuit Dismantle button, eliminate fault
Starter closes when isolating switch is closed ("Start" button untouched) Starter opens when "Stop" button is pushed, but recloses when button is released	Wrong connection of cable conductors at push button station or in starter box	Connect cable conductors according to circuit diagram and terminal arrangement in starter box
Starter fails to open when "Stop" button is pushed	Short circuit between control circuit conductors	Locate and eliminate fault
Starter remains closed only if "Start" button is held down	Open control circuit	Eliminate open-circuit
	"Start" button interlock burned or dirty Cable conductors connected in wrong manner in starter box or push button station	Clean interlock contact surfaces Reconnect cable conductors according to circuit diagram and arrangement of terminals in starter box
Starter contactor hums loudly when closed	Missing or broken shading coil Armature fails to fit close to core on closing due to dents, dirt or mechanical damage Loose coil core Poor contact in "Start" button interlock Excessive spring pressure of moving contacts	Put coil back in position or replace it Smooth down mating surfaces, remove dirt, remedy mechanical defects Tighten fixing screws Wipe contact surfaces clean, adjust contact pressure Re-adjust pressure
Starter opens when high-power motor (coal-cutter motor for example) is started up	Excessive voltage drop across circuit at starting, and coil cannot keep contactor closed	Replace cable in circuit with feeder of greater conductor size, move forward inbye substation

Renewable Fusible Elements for Manual and Magnetic Starters

Only factory-made standard renewable fusible elements may be used in the fuses of manual and magnetic starters. Under no circumstances is it permissible to use make-shift elements made from ordinary wire.

The selection of renewable fusible elements for the links of manual and magnetic starters will be discussed later in Chapter IV, "Underground Cable Circuits".

Table 13

Starter box	Rated current of starter box, A	Type of fuse cartridge	Current rating of fuse cartridge, A	Standard current ratings of factory-made fusible elements, A
380V Boxes				
ПМВ-1357 (PMV-1357)	120	ПР-2 (PR-2)	200	100; 125; 160; 200
ПМВР-1451 (PMVR-1451)	120	ditto	200	100; 125; 160; 200
ПМВ-1344 (PMV-1344)	80	ditto	200	100; 125; 160
ПМВР-1441 (PMVR-1441)	80	ditto	200	100; 125; 160
ПМВ-1331 (PMV-1331)	60	ditto	100	60; 80; 100
ПМВД-1013 (PMVD-1013)	80	ditto	100	60; 80; 100
ПРВ-1007 (PRV-1007)	80	ditto	200	100; 125; 160; 200
ПРВ-1031 (PRV-1031)	60	ditto	100	60; 80; 100
ПБГ-380 (PBG-380)	15	ditto	60	10; 15; 20; 25; 35; 60
ШР-1 (ShR-1)	15	ditto	60	10; 15; 20; 25; 35; 60
660V Boxes				
ПМВ-1357 (PMV-1357)	120	ПР-2 (PR-2)	200	100; 125; 160; 200
ПМВР-1451 (PMVR-1451)	120	ditto	200	100; 125; 160; 200
ПМВ-1344 (PMV-1344)	80	ditto	200	100; 125; 160
ПМВР-1441 (PMVR-1441)	80	ditto	200	100; 125; 160
ПМВ-1331 (PMV-1331)	60	ditto	100	60; 80; 100
ПРВД-1013 (PRVD-1013)	80	ditto	100	60; 80; 100
ПРВ-1007 (PRV-1007)	80	ditto	200	100; 125; 160; 200
ПРВ-1031 (PRV-1031)	60	ditto	100	60; 80; 100
127V Starters				
Starter of АП (ТСШ-2М) [AP (TSSh-2M)] unit	10	МПГ-1 (MPG-1)	10	10

7-7. Push Button Stations

The closing and opening of magnetic starters is carried out by means of push button stations. For mines, the stations are available in four flameproof types, namely:
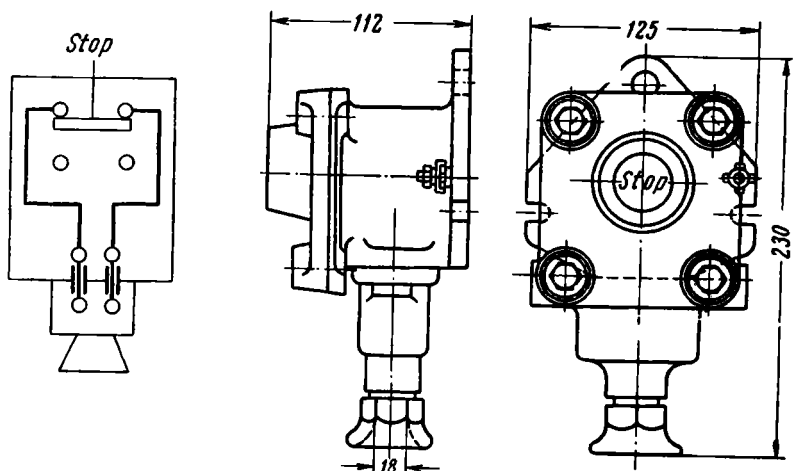
Fig. 203. General view and circuit of КУВ-6011A (KUV-6011A) push button station
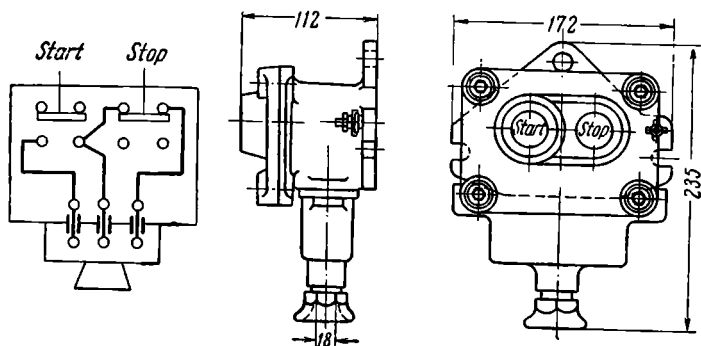
Fig. 204. General view and circuit of КУВ-6012A (KUV-6012A) push button station

(1) КУВ-6011A (KUV-6011A)—a single-button station with a "Stop" button serving for signalling and emergency stopping;
(2) КУВ-6012A (KUV-6012A)—a two-button station with a "Start" and a "Stop" button used to close and open nonreversing magnetic starters;

(3) KУB-6013A (KUV-6013A)—a three-button station with a "Forward", "Reverse" and a "Stop" button to open and close reversing magnetic starters;

(4) KУB-6021A (KUV-6021A)—a two-button station with a "Start" and a "Stop" button used for direct incorporation in a machine enclosure.

The KУB (KUV) push button stations consist of contact elements built into a cast flameproof housing. Each contact element is mounted on an insulating base block and comprises two pairs of fixed contacts (one upper pair and one lower pair) between which a moving contact bridge can move. Pressure on the button first makes the bridge open the upper pair of contacts and then close the lowest pair. The return spring provided in the element always lifts this bridge back on release of the button.

For entrance and sealing of the control cable, each push button station has a sealing-end fitting attached to its housing. The cable conductors in these stations are connected by means of through-type terminals.

Fig. 203 gives a general view and the circuit arrangement of a KУB-6011A (KUV-6011A) push button station; Fig. 204 shows the features of a KУB-6012A (KUV-6012A) push button station.

Chapter VIII

REMOTE CONTROL
OF MINING EQUIPMENT

8-1. General

By *remote control* is meant the operation of equipment through a device stationed some distance from it.

Automatic control is the term used to mean that a unit is controlled by such a system which performs an operation or sequence of operations without human interference.

Below are listed some of the advantages of remote control over local manual control.

1. Safety of operation is greater because the control circuit operates at a low voltage (36V) which is not dangerous to the operator manipulating the controls.

2. The power circuit is interrupted in a starter box set up at some point in a gate road. This eliminates the arcing incident to closing and opening of the starter contactor which would otherwise occur at the face and enhances safety.

3. A magnetic starter incorporates inherent no-voltage protection, i.e., any failure of the voltage will cause it to open. After the voltage reappears, it will re-close only after the "Start" button has been pushed again.

4. Blow out of a power fuse causes the starter contactor to be tripped open. In the case of manual control a motor would continue to operate on the two remaining phases. Very frequently this ends in dangerous overheating of the stator windings.

5. It is possible to incorporate electrical interlocks in the plug-and-socket cable connectors by making the pins of the starter control circuit shorter than the power-circuit contact pins. Hence, when the connector is separated, the first to break circuit with the socket contacts are the control-circuit pins. The starter contactor is thus opened earlier than the power-circuit pins come out of contact

in the socket. In this manner the power circuit is also broken at the starter box.

6. It is possible to attain sequence starting of a conveyor line consisting of several conveyors. If manual control were used, it would be necessary to start the line by operating each manual starter at its conveyor drive station. On occurrence of an emergency in one of the conveyors, it is necessary to immediately shut down the drive motor. This is very difficult to do quickly when manual control is used. Remote control, however, makes it possible to start and stop the entire line automatically, reduce the number of conveyor operators required and at the same time attain greater reliability in operation.

7. In mines not considered as gas and coal-dust hazardous, the cable earthing conductor can be incorporated in the control circuit so as to open the starter contactor the instant the continuity of the earthing circuit is disturbed (earthing circuit monitoring). The electric shock danger is thereby substantially reduced.

Remote control systems may be divided into two groups: those suitable for gas and coal-dust hazardous mines, and those suitable

Fig. 205. Current path in control circuit when a coalcutter runs into a conveyor

for mines free from gas and coal-dust hazards. In the first case the cable earthing conductor cannot be used in the control circuit if the latter is not intrinsically safe. In the second case this is permissible.

Connection of the cable earthing conductor into the control circuit may lead, in certain instances (for example, when a coal cutting machine runs up against a scraper-chain conveyor), to dangerous sparking between metal parts and cause an explosion of the mine atmosphere (Fig. 205). Before the machine parts meet, the current drawn by the contactor operating coil passes through the following path: step-down transformer secondary, contactor coil, contactor interlock, cable conductor, "Stop" button, machine frame, cable earthing conductor, starter enclosure, and step-down transformer secondary. When the machine parts touch, the current follows a second parallel path: coal cutting machine frame, wire rope or

chain, conveyor pans, conveyor motor, conveyor-water cable earth-
ing conductor, conveyor starter enclosure, coal-cutting-machine
starter enclosure, and step-down transformer secondary. The cur-
rent flowing through the parallel path can lead to dangerous spark-
ing at the point where the machines come in touch.

This is the reason why control circuits which are not intrinsic-
ally safe must not use the cable earthing conductor if the mine
is gas and coal-dust hazardous.

8-2. Remote Control of Electric Drills

Remote control of hand electric drills is accomplished with either
ТСШ-2м (TSSh-2m)* transfomer units incorporating two built-in
magnetic starters, or with АП (AP) starter units having intrinsic-
ally-safe control circuits. Fig. 206 gives the electric circuit by
which hand electric drills are remotely controlled in gas and coal-
dust hazardous mines with a ТСШ-2м (TSSh-2m) transformer unit.
The transformer can simultaneously supply two drills through
contactors *K*1 and *K*2 (or one drill and the local lighting circuit).
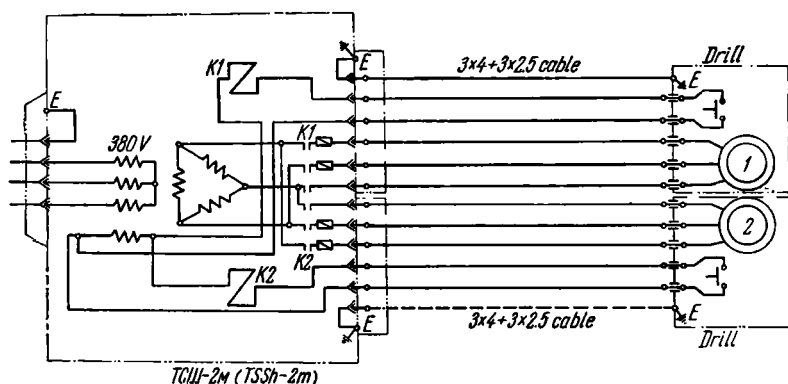
Fig. 206. Remote control circuit for hand-held electric drills operated
from a ТСШ-2м (TSSh-2m) transformer unit

Connection of an electric drill to the transformer is made with a
6-core trailing cable, three of the cores serving for power supply,
two for control and one for earthing. The switch built into the
electric-drill body serves to control the drill by operating the
respective contactor, the drill remaining switched on as long as the
control handle is held down. Fuses inserted in two of the power
phases protect each trailing cable from short circuits.

_____
* ТСШ-2м (TSSh-2m) transformer units are now being produced under the
new type designation ПА (PA), meaning "starting unit",

In mines free from gas and coal-dust hazards, the same circuit can be used for remote control with 5-core trailing cables, the earthing conductors of which are connected into the control circuit. In such cases drill earthing circuit continuity is monitored constantly.

Fig. 207 gives the remote control circuit arrangements for drills operated from an АП (AP) starter unit. Here the control circuit of the unit is intrinsically-safe and the trailing cable earthing conductors are hence incorporated in the control circuit. The unit is supplied at 380 or 660 volts through a ПМВ-1331 (PMV-1331) magnetic starter box. Earth leakage protection in the 127-volt circuit is provided by the РУВ-2-127 (RUV-2-127) earth leakage relay connected to the starter unit and arranged to open the magnetic starter.

Included within the starter unit are: the power step-down transformer Tr_1 (for supply of two electric drills) with a primary that can be either star-connected (for 660-volt supply) or delta-connected (for 380-volt supply), a 133/36-volt step-down transformer Tr_2 for control circuit supply, two line contactors LC_1 and LC_2, two auxiliary relays AR_1 and AR_2 fed from selenium rectifiers SR_1 and SR_2, and two overcurrent relays OCR_1 and OCR_2 for protecting the trailing cables from short circuits.

When the control-switch lever on a drill is pressed (CS_1, for instance), auxiliary relay coil AR_1 is energized by the current flowing through the circuit: from top terminal of Tr_1 transformer secondary considered to be at positive potential at the moment; arm a of bridge-connected rectifier SR_1, auxiliary relay coil AR_2, arm b of rectifier SR_1, closed overcurrent relay contact OCR_1, drill control switch lever DC_1, cable earthing conductor, starter unit enclosure and midpoint of Tr_2 transformer secondary. As soon as the polarity of the transformer terminals reverses, auxiliary relay coil AR_1 is fed without change in direction by the other pair of arms in rectifier SR_1. Auxiliary relay AR_1 therefore picks up, closes its AR_1 contact in the LC_1 contactor coil circuit and thereby closes the contactor to apply power from the Tr_1 transformer secondary to the drill motor. In the event of a short circuit in the cable, overcurrent relay OCR_1 operates to open its OCR_1 contact in the supply circuit of auxiliary relay AR_1. The latter is de-energized, and the drill motor is disconnected from the supply.

Post-mounted electric drills are connected to the circuit through a ПБГ-380 (PBG-380) manual starter box or CM-380 (SM-380) cable coupler. The drill motor is started, stopped and reversed by a switch built into the drill housing. Remote control is accomplished with an АПК (APK) starter unit. When the latter is employed, the control circuit is the same as that given for hand electric drills.
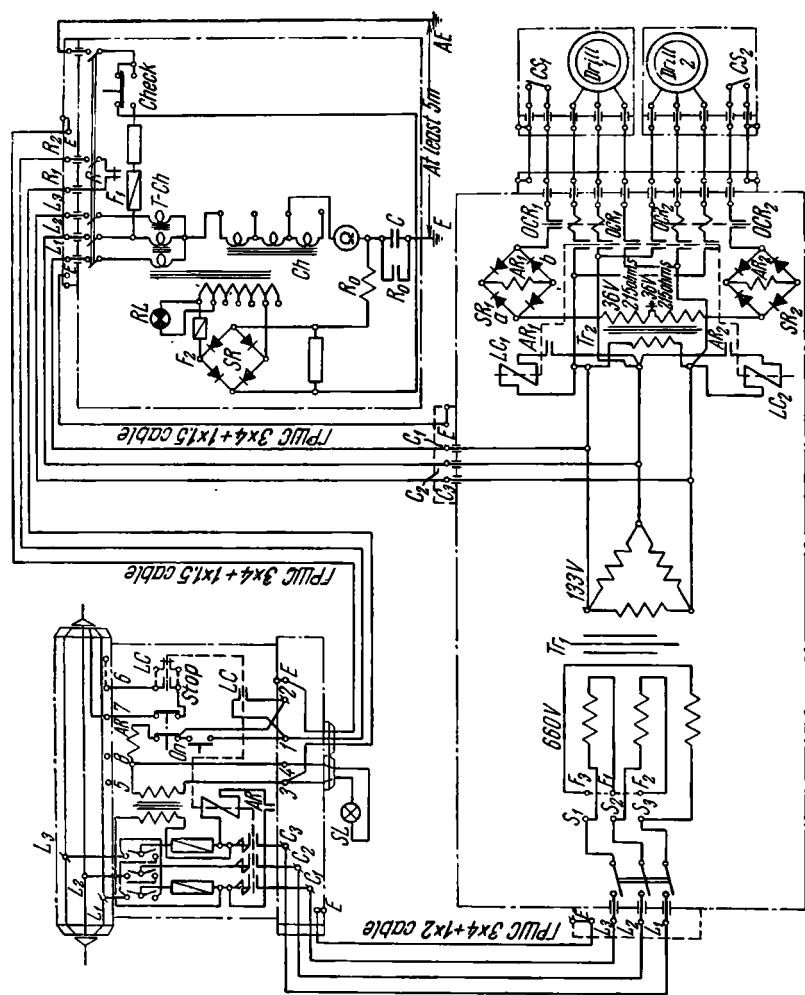
Fig. 207. Remote control circuit for hand-held electric drills operated from an АП (AP) starter unit

8-3. *Remote Control of Coalcutters and Cutter-loaders*

A coal cutting machine is controlled through the "Start" and "Stop" (Fig. 208) buttons built into its frame. The controller also built into the machine has an interlock which is connected in the starter control circuit and is operated by the controller shaft. The interlock closes only after the controller power contacts make when the controller is closed, and opens only after the break when the controller is open. This sequence precludes any making or breaking of a live power circuit by the controller. This is done by the starter box situated in the gate road. As soon as the controller is closed and the "Start" button is pushed, the starter box auxiliary relay AR is energized by the current flowing through the path: Tr transformer secondary, auxiliary relay coil AR, starter "Stop" button, cable control conductor, cutting-machine "Start" and "Stop" buttons, controller interlock CI, second control conductor of cable, and transformer secondary. Auxiliary relay AR picks up to close its AR contact in the LC line-contactor operating coil circuit. Line contactor LC then closes to supply power to the motor circuit through its main contacts and also shunt the "Start" button with its interlock LC. Study of the circuit makes it evident that the auxiliary relay circuit is completed through the cable earthing conductor and thereby keeps it constantly under control for continuity. Occurrence of a break in the earthing conductor or a substantial rise in resistance in the contact connections will make auxiliary relay AR drop out, open its contact in the LC coil circuit, and open the starter box line contactor.

Pressure on the "Stop" button or opening of the controller de-energizes auxiliary relay AR to make the starter contactor trip open.

Fig. 209 gives the control circuit of a coal cutting machine to be operated in a gas and dust-hazardous mine. Here the earthing conductor can serve only for earthing, while two other cable conductors serve for control, for which purpose the resistor R shunts the "Stop" button and limits the current in the circuit to a value at which the auxiliary relay AR will not pick up if the "Start" button contact is open. When a start is initiated by pushing the "Start" button, resistor R is shunted out of circuit, the current rises to a value at which auxiliary relay AR picks up, and the starter box contactor closes. The instant the "Start" button is released, the resistor is again brought into circuit and the current in the circuit drops to its original value. Since the relay is closed, the current remains sufficient to maintain it picked up. The starter contactor is opened by pressure on the "Stop" button or opening of the controller interlock CI.
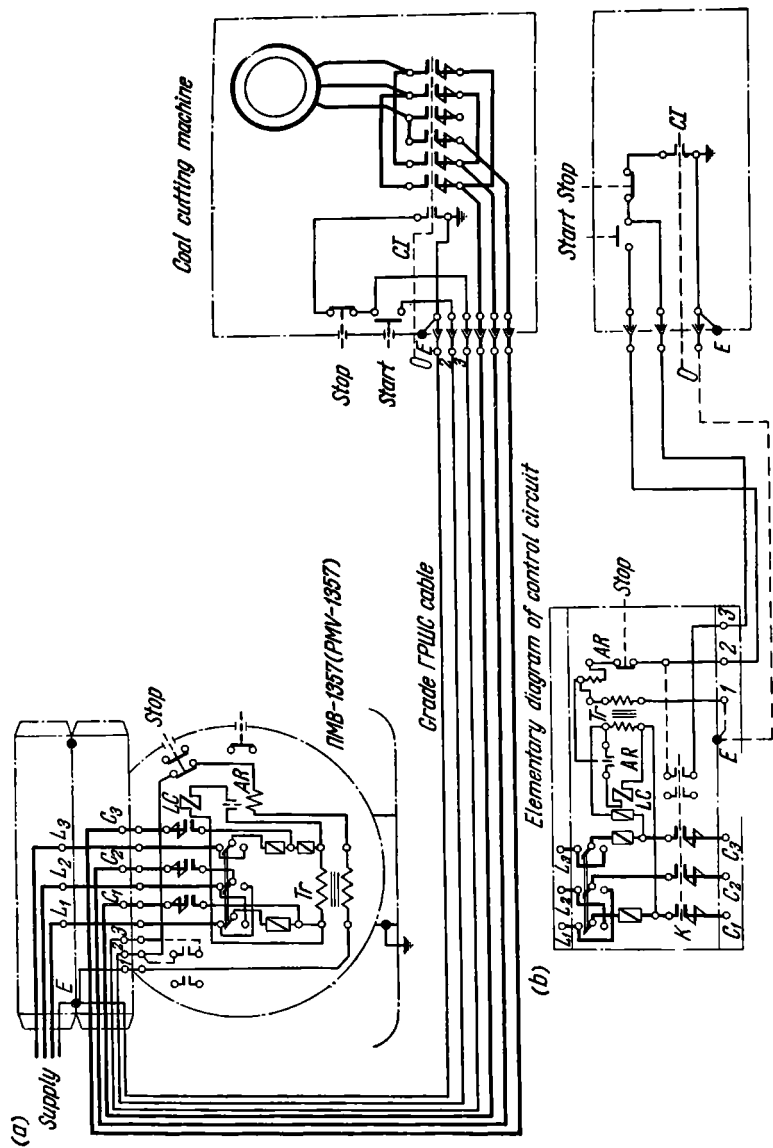
Fig. 208. Coalcutter remote control circuit for mines free from gas and dust hazards:
a—wiring diagram; *b*—elementary diagram

Fig. 209. Coalcutter remote control circuit for gas and dust hazardous mines:

a—wiring diagram; *b*—elementary diagram
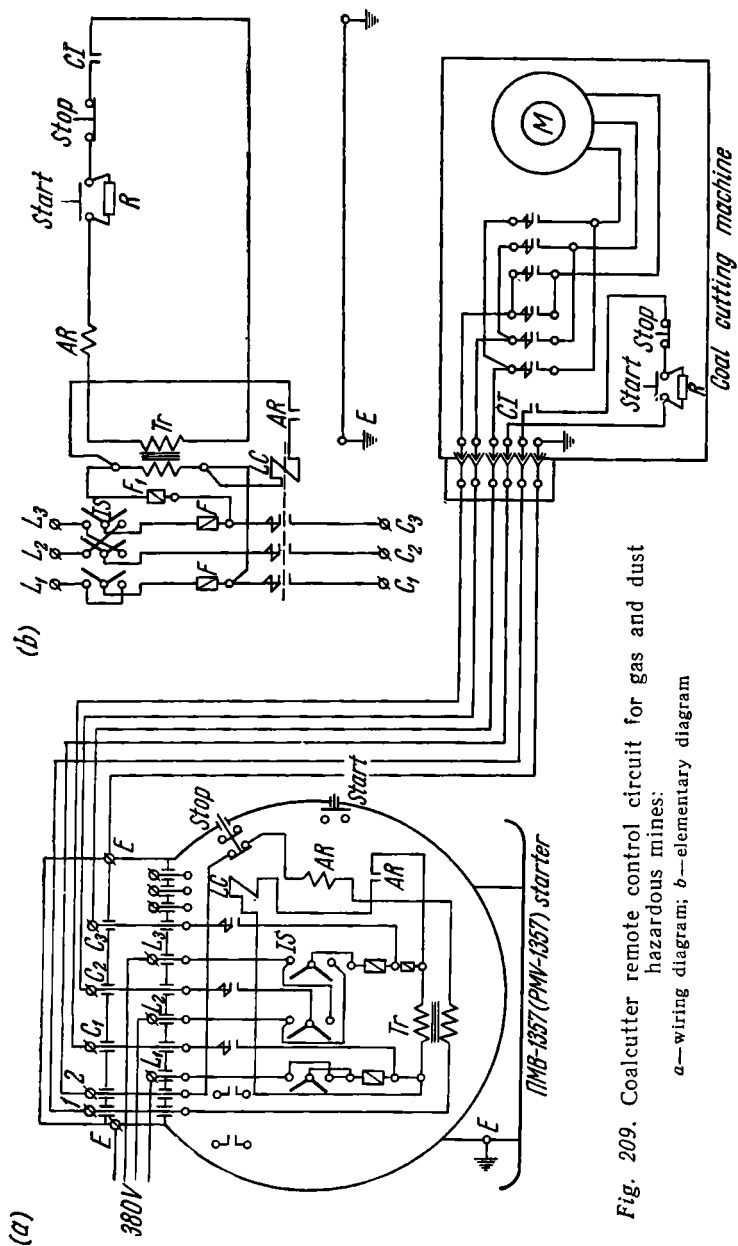
In both of the above circuit arrangements, the starter contactor will automatically open on occurrence of an excessive drop in the supply voltage (undervoltage protection) or a short circuit in the power circuit.

Magnetic starter boxes with intrinsically-safe control circuits make it possible to employ remote control schemes which provide a continuous check on the continuity of the earthing conductor within gas and coal-dust hazardous mines. The earthing conductor in such circuits simultaneously functions as a control conductor.

Fig. 210 shows the remote control circuit of a КМП-3 (KMP-3) coal cutting machine in which a ПМВР-1451 (PMVR-1451) reversing magnetic starter is included. The latter has an intrinsically-safe control circuit. Starter box control is achieved by three buttons, "Forward", "Reverse" and "Stop", connected through the three control conductors of the trailing cable. The starter contactor can also be opened by the "Stop" button provided directly on its enclosure. Auxiliary relays $AR1$ and $AR2$ have their coils shunted by diode rectifiers $D3$ and $D4$. At the push button station on the machine, two other rectifiers $D1$ and $D2$ are included.

Supply for auxiliary relays $AR1$ and $AR2$ in the starter is taken from winding II of the starter step-down transformer Tr.

Consider the above winding when its "a" terminal is positive and its "b" terminal negative. If now the "Forward" button is pushed, the $AR2$ relay coil will be energized by current through the following circuit: from transformer Tr_5 terminal "a", through rectifier $D3$*, auxiliary relay coil $AR2$ (no path for current exists through $D4$ in this direction), normally-closed contactor interlock $K1$, terminal 2, "Forward" push button, rectifier $D1$, "Stop" button, interlock button, earth terminal E, earthing conductor, starter "Stop" button, and transformer winding II terminal "b". During the negative half-cycle the polarity in winding II is reversed. However, the path for current through the earthing conductor, the interlocking push button and the machine "Stop" button from positive terminal "b" is blocked by rectifier $D1$ which cannot pass current in the reverse direction. Consequently, the current will flow through the relay coil $AR2$ only during one half of the cycle (when terminal "a" is positive), i. e., relay $AR2$ operates on half-wave rectified current supply. This is sufficient to make it pick up and close its $AR1$ contact in the $K2$ contactor coil circuit. If contactor $K1$ is de-energized, its normally-closed interlock in the $K2$ contactor control circuit is

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* Since terminal a at this instant is positive, the current flows through $D3$ (as indicated by the arrow) and by-passes relay coil $AR1$ which has a much greater resistance at this moment than rectifier $D3$.
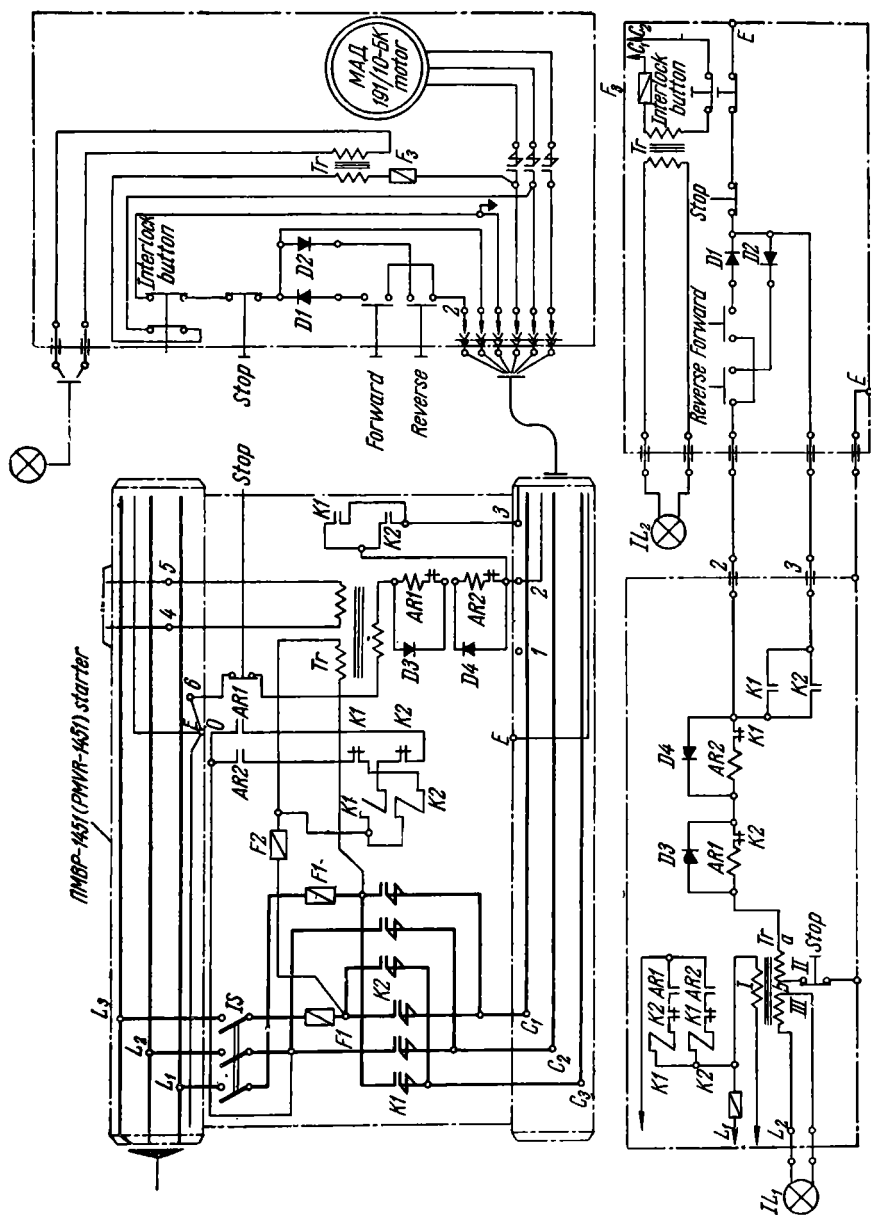
Fig. 210. Remote control circuit for КМП-3 (KMP-3) coalcutter
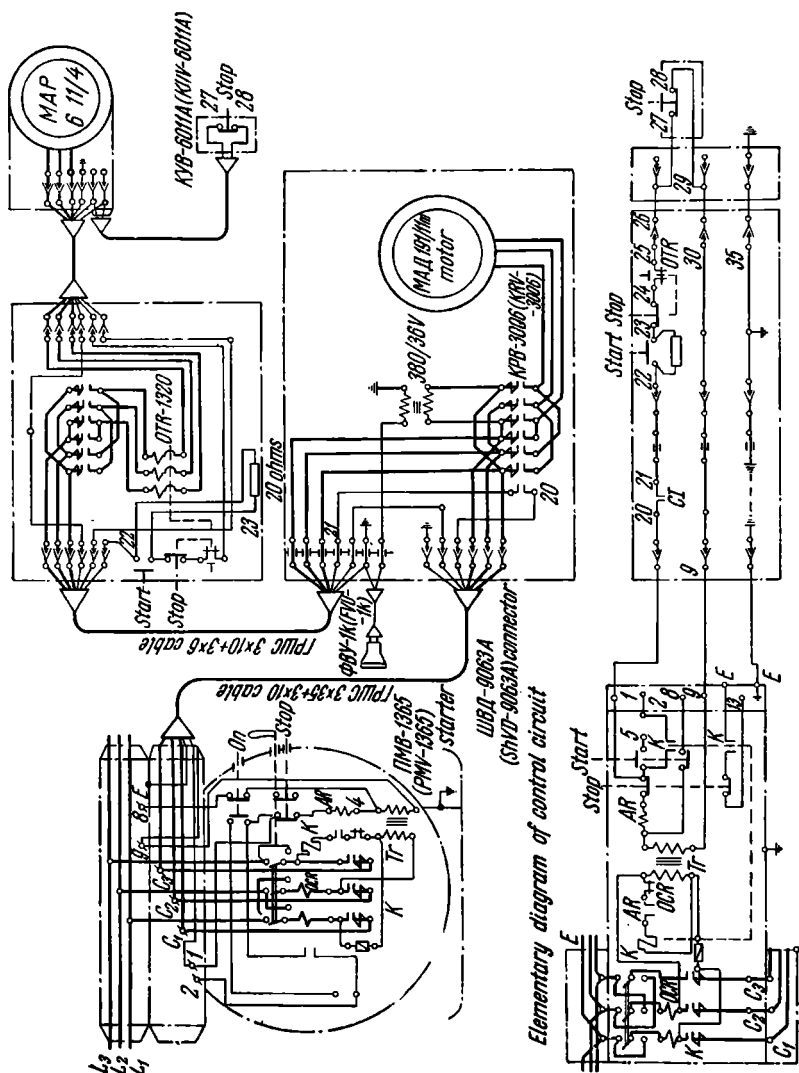
Fig. 2/1. Remote control circuit of a Donbass-1 cutter-loader for operation in mines free from gas and dust hazards

closed and the contactor $K2$ will then close. Interlock $K2$ in the circuit of auxiliary relay coil $AR1$ opens simultaneously to prevent closing of contactor $K1$. Normally-open interlock $K2$, on the contrary, closes to allow the $AR2$ auxiliary relay to remain energized round the "Forward" button.

Pressure on either of the "Stop" buttons in the circuit restores it to its initial (de-energized) condition. Following this, contactor $K1$ can be closed by pressure on the "Reverse" button. If terminal "a" of winding II is positive at this instant, no current will flow through auxiliary relay coil $AR1$ because rectifier $D2$ is non-conducting in this direction. When the next half-cycle begins, terminal "b" becomes positive and auxiliary relay coil $AR1$ is energized through the path: terminal "b", "Stop" button in the starter, cable earthing conductor, machine interlock "Stop" buttons, rectifier $D2$ "Reverse" button, cable control conductor, terminal 2, rectifier $D4$, normally-closed interlock $K2$, auxiliary relay coil $AR1$, and terminal "a". Auxiliary relay $AR1$ is thereby energized by a half-wave rectified current sufficient for its operation, makes its contact, and closes the contactor $K1$.

Fig. 211 gives the remote control circuit arrangement of the Donbass-1 coal cutter-loader for mines free from gas and coal-dust hazards. The trailing cable earthing conductor is connected in the control circuit. Control of the main motor in the machine is accomplished from the "Start" and "Stop" button stations built into the loader-motor controller enclosure.

A manual reversing controller serves to operate the loader motor. The motors of the machine may also be stopped by the loader "Stop" button, provided with a latch to lock the button when the cutter or cleaver picks are being replaced. The motors are switched on in the following order: with haulage feed cut out and the loader-motor controller opened, the main motor controller is closed. Then, pressure on the "Start" button closes the magnetic starter to feed the main motor. After the main motor has set into run, the loader motor can be started with its controller. Simultaneous starting of both motors would cause overheating to the supply cable, starter and incoming-cable plug-and-socket connector. The loader motor has a built-in overcurrent and thermal relay OTR, operation of which trips the magnetic starter open. To reclose the starter after an operation of the above relay, the latter must first be re-set. This is done by pressure on the "Stop" button on the machine.

The remote control circuit arrangement of the Donbass-1 coal cutter-loader for gas and coal-dust hazardous mines may be seen in Fig. 212. The cable earthing conductor is excluded from the control circuit, and the "Start" button is shunted by a resistor.

12*
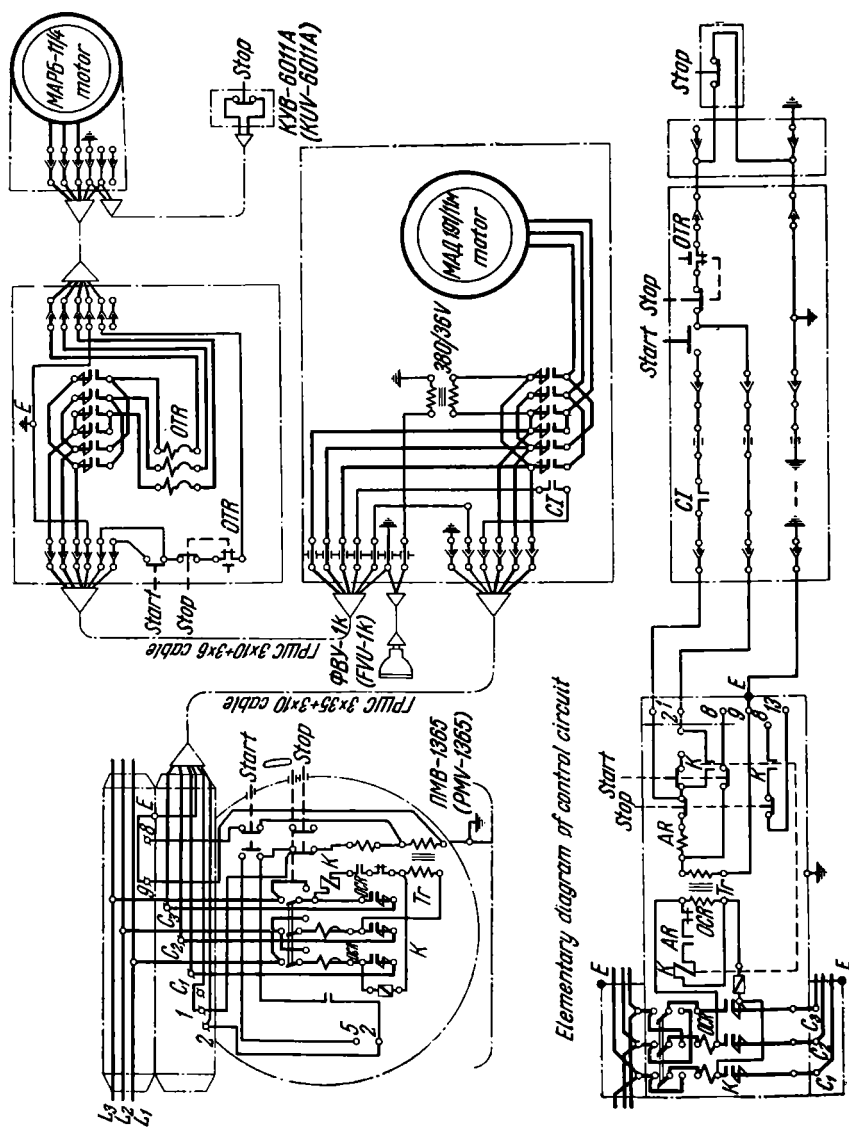
Fig. 212. Remote control circuit of Donbass-1 cutter-loader for operation in gas and dust hazardous mines

8-4. *Remote Control of Heading Machines*

Heading machines incorporate several motors for driving their various mechanisms. Their remote control circuits therefore possess certain particular features.

Fig. 213 *a* and *b* shows the control circuits of a ПК-2м (PK-2m) cutter-loader for use in gas and coal-dust hazardous mines. Supply from a gate-end ПМВ-1357 (PMV-1357) magnetic starter box is fed to the magnetic starter (also a PMV-1357 box) mounted on the machine. This second starter feeds the main motor *1* and the machine distribution board. To close the gate-end magnetic starter, the "On" button of station *PB* must be pushed; this button is shunted with a resistor. Included in the gate-end starter control circuit are two control switch interlock contacts *CS*-1 and *CS*-2, closed when the control switches are in their "Off" positions, and three distribution board and cover interlock buttons *PB*4, *PB*5 and *PB*6. The cable earthing conductor is not used in the control circuit. The ПМВ-1357 (PMV-1357) magnetic starter box mounted on the machine is operated by push button station *PB*1. Belt-conveyor motor *2* is controlled by contactor *K*6 and push button station *PB*2, while control of crawler-drive motors *3* is accomplished through control switches *CS*-1 and *CS*-2 built into the control panel.

With the main drive operating and control switch *CS*-1 set in its "Auto-advance" position, pressure on auto-advance push button *PB*3 establishes a current path through closed contacts *CS*-1-7(*C*) and *CS*-1-8(*C*), and interlocking contact *CS*-2-7(*C*), to close forward contactors *K*2 and *K*4. When the auto-advance pusher reaches its limit stop, push button *PB*3 opens its contacts to stop further advance into the face. Forward travel is also accomplished with control switch *CS*-1 by setting it in its "Forward" position. The switch contacts *CS*-1-5*B*, *CS*-1-1(*A*) and *CS*-1-3(*A*) then make to reclose contactors *K*2 and *K*4. Reverse travel is achieved by turning control switch *CS*-1 to its "Reverse" position, thereby making contacts *CS*-1-2(*A*) and *CS*-1-4(*A*) and closing contactors *K*3 and *K*5. For making a left or right turn, or a full turn-about, control switch *CS*-2 is provided on the control panel. It is operated when control switch *CS*-1 is in its "Off" position.

By turning switch *CS*-2 to the "Right Turn" position, contact *CS*-2-1(*A*) makes to close the contactor *K*2 of the left-hand track motor. Turning the switch to the "Left Turn" position makes contact *CS*-2-3 (*A*), thereby closing the contactor *K*4 of the right-hand track motor. To accomplish a right-hand full turn-about, the left-hand track motor is switched on for forward travel and the other motor is switched on for reverse travel. Switch contacts *CS*-2-1(*A*) and *CS*-2-4(*A*) must make to close contactors *K*2 and *K*5 for this operation. For

Fig. 213. Control circuits of ПК-2м (PK-2m) heading cutter-loader:

a—wiring diagram; b—elementary diagrams
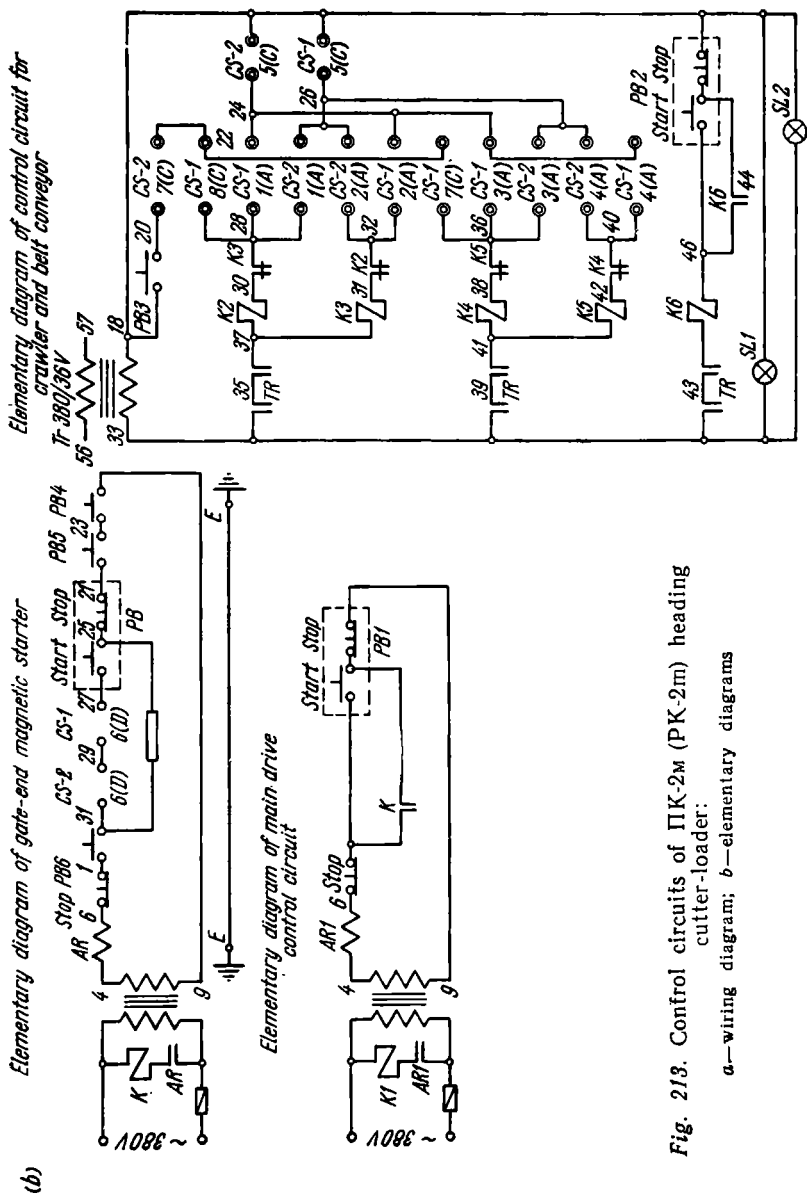
a left-hand turn-about, contacts *CS-2-2(A)* and *CS-2-3(A)* must make to close contactors *K*3 and *K*4.

The contactors of the crawler-track motors are also mechanically interlocked with each other (*K*2 with *K*3 and *K*4 with *K*5).

To prevent the track motors from being simultaneously switched on for both turns (turn-abouts) and travel in the forward or reverse direction, both control switches *CS*-1 and *CS*-2 are interlocked with each other.

Fig. 214 shows the elementary power and control circuit diagrams for the ПK-3 (PK-3) cutter-loader. Mounted on this machine is a magnetic controller box which receives its supply from a gate-end ПMB-1357 (PMV-1357) magnetic starter box. The latter is operated by the isolating contactor interlock *IC* on the magnetic controller. To close the starter box contactor, control switches *CS*-1 and *CS*-2 must be put in the "Stop" position to make their contacts *7* and *8*. When the starter contactor is closed, resistor r_1 shunts the above contacts to hold in starter auxiliary relay *AR*. Contactors *K*1 and *K*3 (on the magnetic controller board) serve to control the main drive (the cutting head unit) and oil pump motors with the aid of the *PB*1 push buttons. Conveyor and loader motor control is effected by means of contactors *K*2 and *K*4, and the *PB*2 push buttons.

The crawler-track motors are controlled by contactors *K*5, *K*6, *K*7 and *K*8 (forward, reverse), and control switches *CS*-1 and *CS*-2. Each pair of "crawler-track" contactors is interlocked by the respective normally-closed contactor interlocks, *K*5, *K*6, *K*7 and *K*8.

This control scheme permits the machine to travel forward, backwards, make a left-hand or right-hand turn, and turn about to the right or left.

Control switch *CS*-1 has three fixed positions. In the "Stop" (off) position, contacts *5-6* and *7-8* make; in the "Forward" position contacts *1-2* and *3-4* make, and in the "Reverse" position contacts *9-10* and *11-12* make.

Control switch *CS*-2 also has three fixed positions. In the "Stop" position, contacts *5-6* and *7-8* close; in the "To Right" position contacts *3-4* close. Contacts *2* and *3-4* close in the "Right Turn" position. Setting in the "To Left" position closes contacts *9-10*, while setting in the "Left Turn" position closes contacts *9-10* and *11*. For protecting the main drive motor from excessive currents which arise when the cutting head stalls during cutting, an over-current relay *OCR*1 is incorporated to open contactor *K*1 by means of the interlocking relay *IR*. Thermal relays *TR* are incorporated for overload protection.
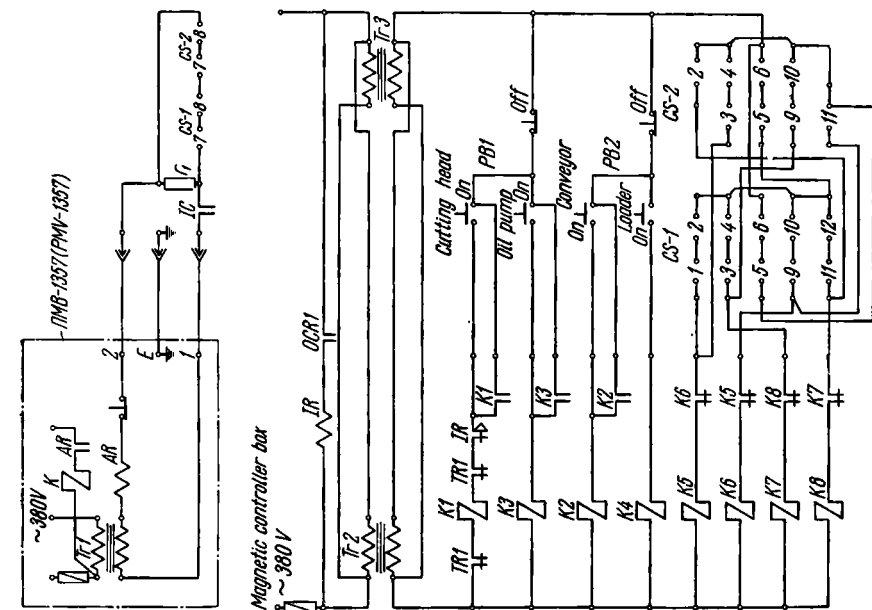
Fig. 214. Elementary power (*a*) and control circuits (*b*) of ПК-3 (PK-3) cutter-loader:

1—cutting-head electric drive motor; *2*—scraper conveyor motor; *3*—oil pump motor; *4*—belt loader motor; *5*—left crawler track motor; *6*—right crawler track motor

8-5. Remote Control of Coal-loading and Rock-loading Machines

A loading machine is connected to the supply through a ПМВ-1344 (PMV-1344) or ПМВ-1357 (PMV-1357) magnetic starter box controlled by a ШБГ-380 (ShBG-380) plug-and-socket switch (Fig. 215). A shortcoming of this circuit is lack of undervoltage protection. It is therefore necessary to switch off the ШБГ-380 (ShBG-380) every time the voltage disappears. The cable earthing conductor is not incorporated in the control circuit in this set up.

In mines free from gas and coal-dust hazards, use is made of a conventional "Start-stop" push button control circuit with the cable earthing conductor included in the control circuit.

Fig. 216 gives the remote control circuit of a ГН.Л-30 (GNL-30) coal-loading machine. Mounted on the magnetic controller of this machine are: a set of contactors $K1$, $K2$, $K3$, $K4$ and $K5$ for control of the gathering unit, loading conveyor and crawler-track motors. Power for the magnetic controller is taken from a ПМВ-1344 (PMV-1344) magnetic starter box at a gate end. The box is controlled from a portable push button station. Its "On" button has a latch, released when the starter is de-energized. The circuit does not include undervoltage protection and a check on the continuity of the earthing circuit. The "Load" push button on the control panel and contactor $K1$ serve to operate the gathering arm and loading conveyor motors. The crawler-track motors are controlled with the "Forward", "Reverse", "Left", and "Right" push buttons which operate two pairs of interlocked contactors $K2$, $K3$ and $K4$, $K5$.

In Figs. 217 and 218 can be seen the wiring and elementary diagrams of the ЭПМ-1 (EPM-1) rock-loading machine control circuit used in gas and coal-dust hazardous mines. This machine takes its power from a gate-end ПМВ-1344 (PMV-1344) magnetic starter through a trailing ГРШ (GRSh) $3 \times 16 + 3 \times 10$ mm² cable. The starter is closed by pressure on the "On" button provided on its enclosure. It is opened by the "Stop" buttons on the machine control panels or by the "Stop" button on the starter box.

The control circuit of the box includes a pair of "Interlock" buttons located in the left-hand and right-hand control panel enclosures. These buttons open the circuit when a panel access cover is removed and thereby open the starter contactor. The right-hand and left-hand control panels comprise a pair of КТД-2АЗ (KTD-2A3) contactors and a set of push button stations. "Forward" and "Reverse" push buttons serve to operate the wheel-drive (traction) motor. These contactors ($K3$ and $K4$) are so interlocked that the motor cannot be reversed when it is running.

Fig. 215. Remote control circuit of coal and rock loading
machines operated in gas and dust hazardous mines:
a—wiring diagram; *b*—elementary control circuit diagram
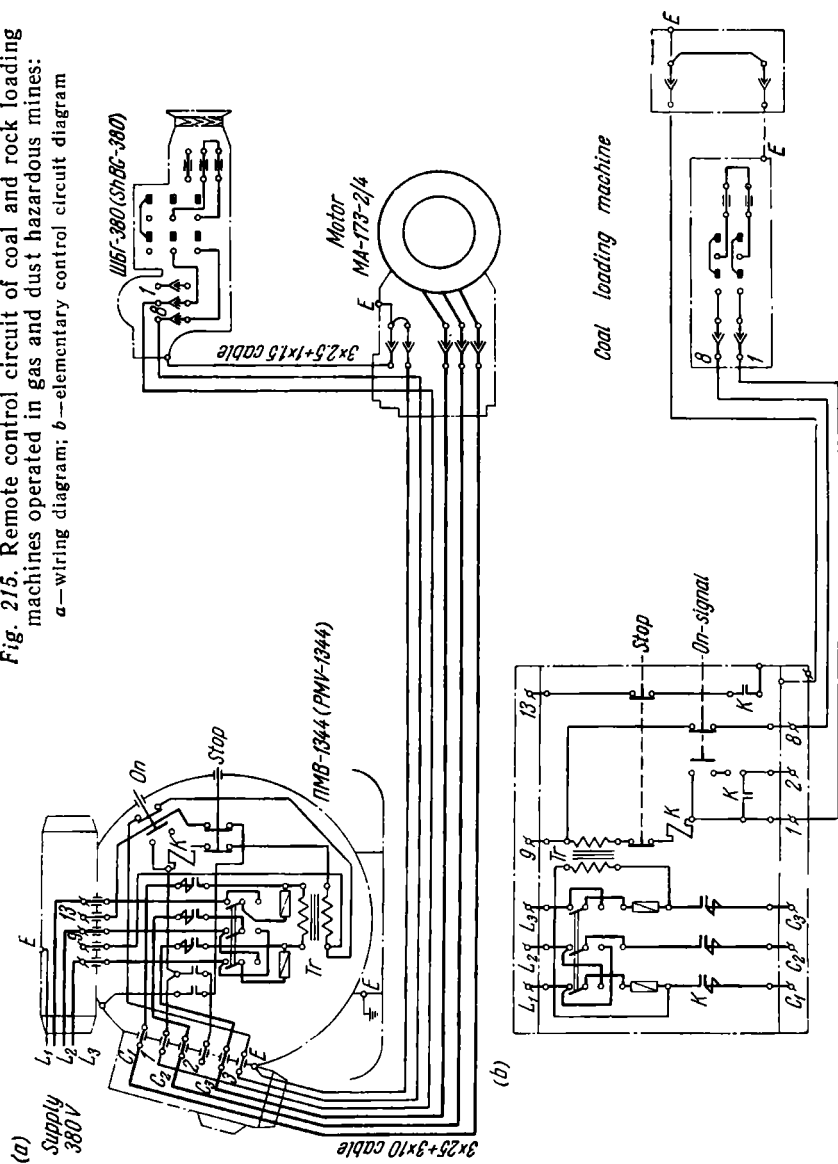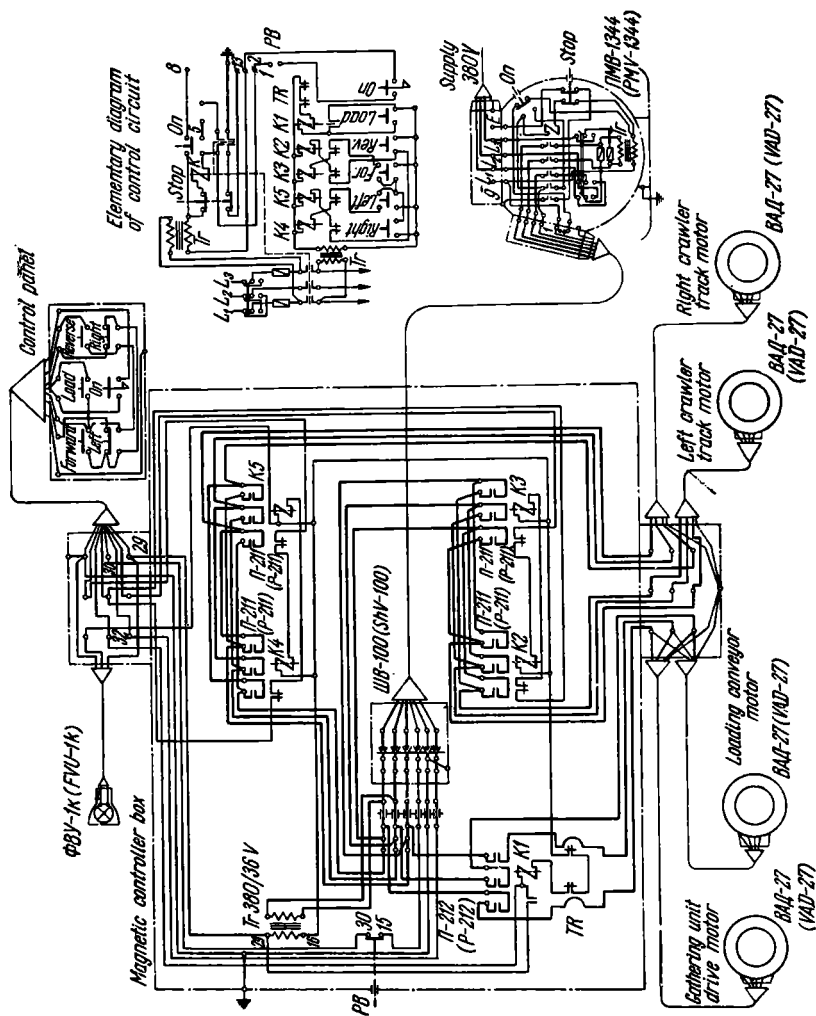
Fig. 216. Remote control circuit of ГНЛ-30 (GNL-30) coal loading machine
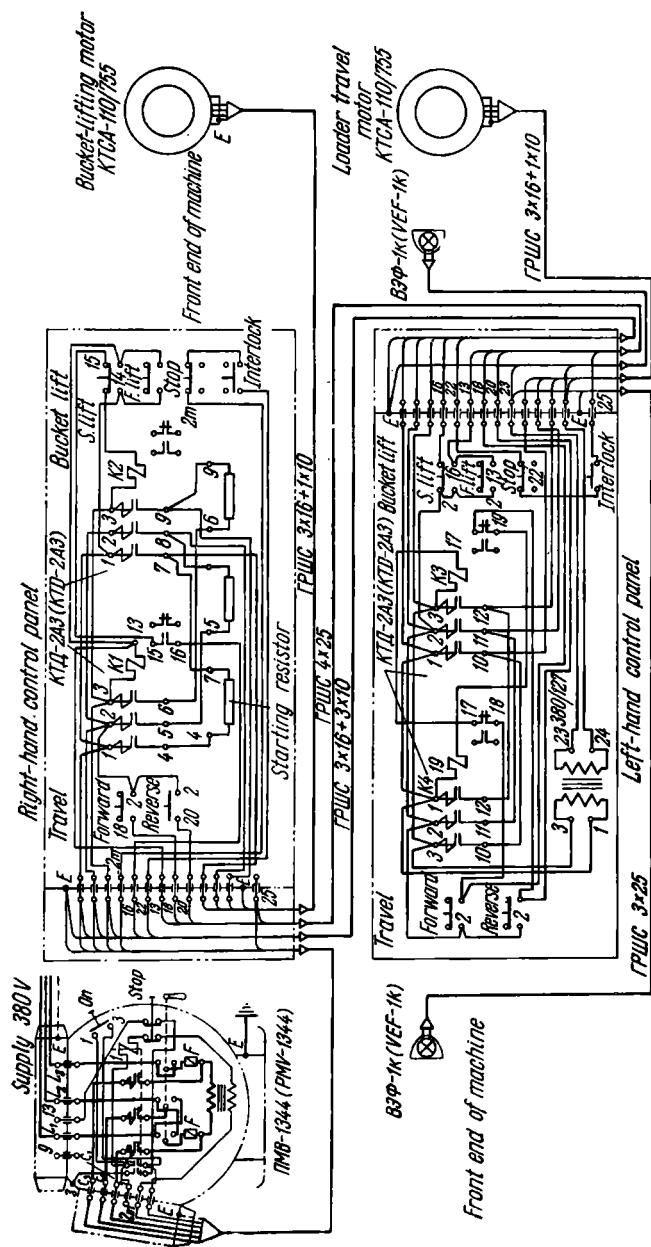
Fig. 217. Wiring diagram of ЭПМ-1 (EPM-1) rock loading machine

The bucket-lifting motor is switched on by means of the "S (slow) lift" and "F (fast) lift" push buttons. Pressure on the "S-lift" button closes contactor $K1$ to connect the bucket-lifting motor to the supply
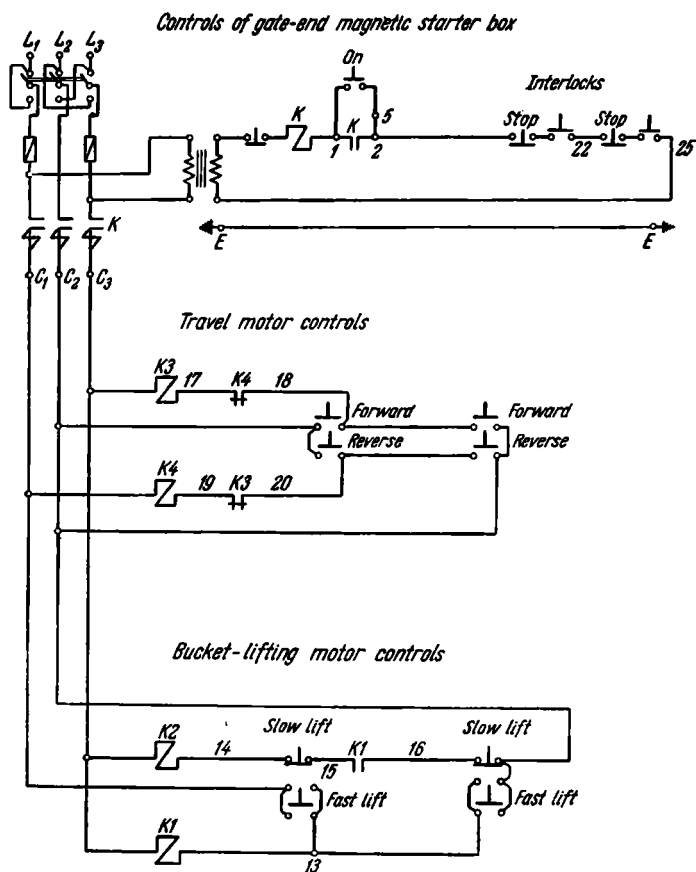
Fig. 218. Elementary control diagram of ЭПМ-1 (EPM-1) rock loading machine

through a set of resistors connected in series with the stator winding. By jogging control of the motor with this button, better bucket filling is attained. Contactors $K1$ and $K2$ by which the motor is slow or fast operated are interlocked by an auxiliary contact $(K1)$ on contactor $K1$. Therefore, when the "F-lift" button is pressed, it first closes contactor $K1$ to switch on the motor through the series resistance; following this it closes contactor $K2$.
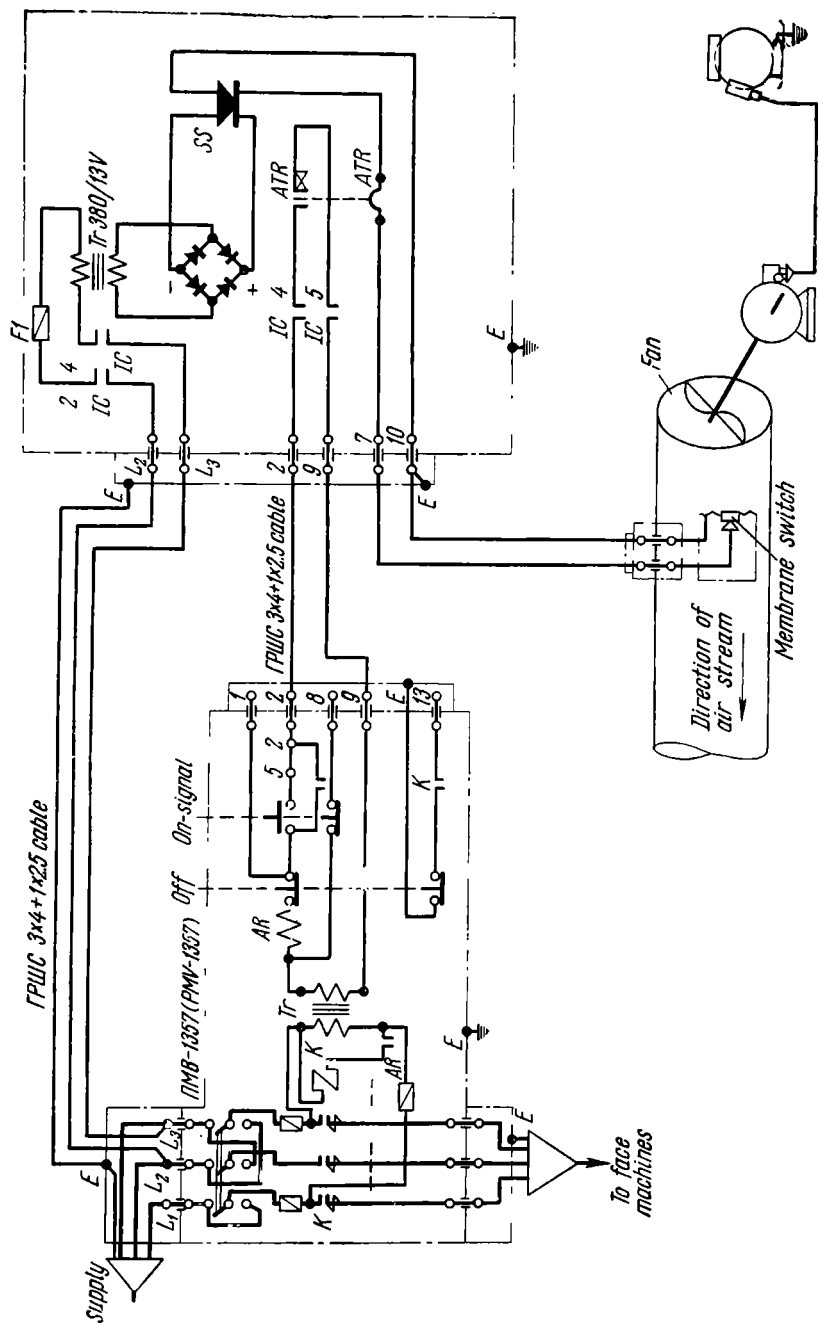
Fig. 219. Remote control and monitoring circuit of a booster fan

8-6. Remote Control of Booster Fans

The principal means of preventing a dangerous mine atmosphere condition from developing at the blind faces of headings is to continuously keep them well ventilated. Owing to this, monitoring of booster or auxiliary ventilating fan operation is given serious attention (Fig. 219).

At a normal air stream velocity in the ventilating duct the membrane switch placed in the air stream closes the circuit of auxiliary thermal relay *ATR*. The latter becomes heated, closes its contacts *1C* and completes the circuit to starter box auxiliary relay *AR*. This makes the starter box contactor close to feed power to the face machines. If the air stream velocity falls below 3.5 metres per second, the membrane switch opens its contacts in the *ATR* relay circuit, the relay cools down and opens its contacts in the starter control circuit to discontinue power supply.

Thermal relay *ATR* can be seen to receive power from transformer *Tr* through a selenium bridge-connected rectifier and selenium shunt *SS*, thereby providing an intrinsically-safe relay circuit.

8-7. Remote Control of Car-spotting Winches

Depending on how cars or tubs are handled, spotting winches will either be of the single- or double-drum type.

To remotely control single-drum winches, which pull up a coal car string for loading, use is made of conventional circuits employing a ПMB-1331 (PMV-1331) or ПMB-1344 (PMV-1344) magnetic starter box and a two-button control station. Double-drum winch control is accomplished with a reversing magnetic starter box [ПMBP-1441 (PMVR-1441), ПMBP-1451 (PMVR-1451)] and a three-button control station (Fig. 220).

8-8. Remote Control of Automatic Pumping Installations

In automatic pumping installations, sump water level monitoring and automatic start-up of pump units is generally accomplished with float-type relays. The latter find wide application in automatic control of drainage system pumping stations and serve to start and stop the pumps, and also give an alarm signal when the water level rises too high.

An automatic drainage pumping installation (Fig. 221) will comprise an airtight priming tank *1*, a float and its rod *2*, free to move upwards and downwards in guides, a switching device *3*, a magnetic starter box *4*, and a pump *5* driven by a motor *6*. The switching device contains two contacts: the first contact closes when the water rises to the normal "start pumping" level; the second
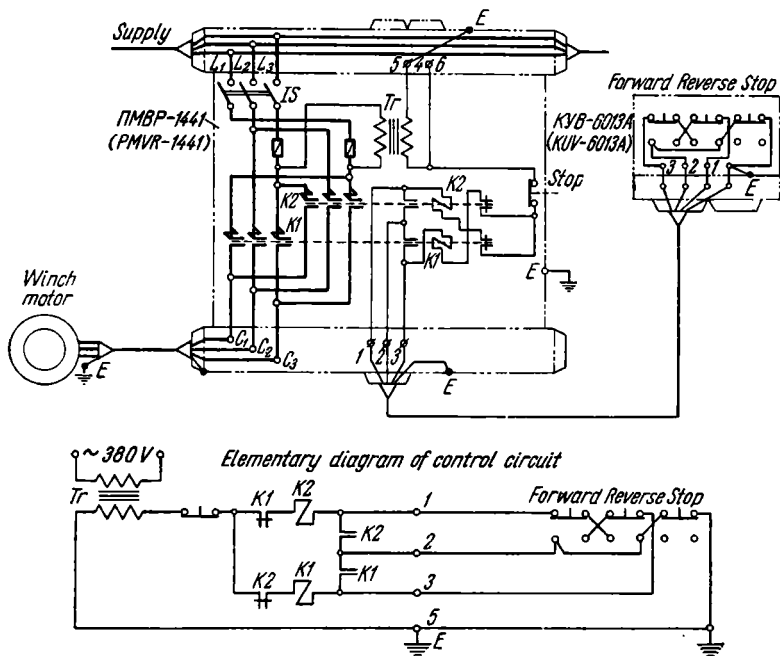
Fig. 220. Remote control circuit of double-drum haulage winch
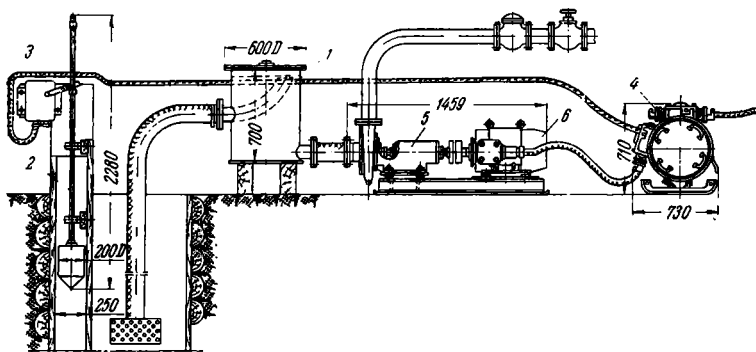
Fig. 221. Arrangements of automatically controlled pump installation

contact closes when the level becomes excessive and its purpose is to start up a standby pump, or energize an alarm device. Movable stops on the float rod set the levels at which the contacts will be closed. When the water reaches a preset level, the rod stop turns the switching device arm upward to close the first contact. Being connected in the starter control circuit, this contact starts up the

pump. If the water level still continues to rise, the float rod also rises to a higher position at which the second contact connected into the alarm energizing circuit is closed.

When the water drops to the desired level, the float simultaneously lowers the rod to make the upper stop bear down on the switching device arm and thereby open the pump control contact and switch off the pump motor.

For the automatic control of water drainage installations (main, auxiliary, district or transfer) containing two or three pumping units powered with 380 V motors of up to 120 kW total rating. Type ABH-1 (AVN-1) automatic control apparatus is available.

Fig. 222. Schematic diagram of pump priming by means of accumulator tank:

1—priming accumulator tank; *2*—suction pipe; *3*—bypass branch; *4*—check valve in delivery pipe line

The ABH-1 (AVN-1) automatic control apparatus provides:

(a) start-up of one of the pumps at a selected high-water level in the sump and shut down at the low-water level, with automatic priming from a priming accumulator tank (Fig. 222) or from a forced-flow standpipe through a controlled valve;

(b) definite operation sequence of the two or three pumps;

(c) consecutive starting (with a slight time delay) of two pumps at a dangerous high-water level;

(d) remote start-up of a pump from the annunciator or lamp register;

(e) check on the condition of the pump hydraulic system with a flow-rate relay placed in a horizontal run of the suction line of each pump.

The above flow-rate relay is constructed with two pressure chambers separated by a membrane. This membrane is linked by a system of levers to a contact assembly. On the horizontal run of the suction line two pressure pick-up tubes are inserted, with one tube-end turned upstream and the other downstream. These pick-up tubes are connected by pipe unions and tubing to their corresponding relay chambers. When water flows through the suction line, the pressure increases in one chamber (connected to the upstream-faced tube) and decreases in the other and the flexible membrane deflects to act upon the contact assembly.

A check on sump water level is provided by a pneumatic water-level relay whose operation is duplicated by a float relay. The pneumatic relay is a bell whose openings in the lower part are submerged in the sump (Fig. 223). As the water level in the sump rises to a certain height, the air within the bell is compressed

to operate the membrane-type pneumatic relay unit connected to the bell by rubber tubing. When the pump lowers the water level to just below the bell ports, the air in the bell is replenished.
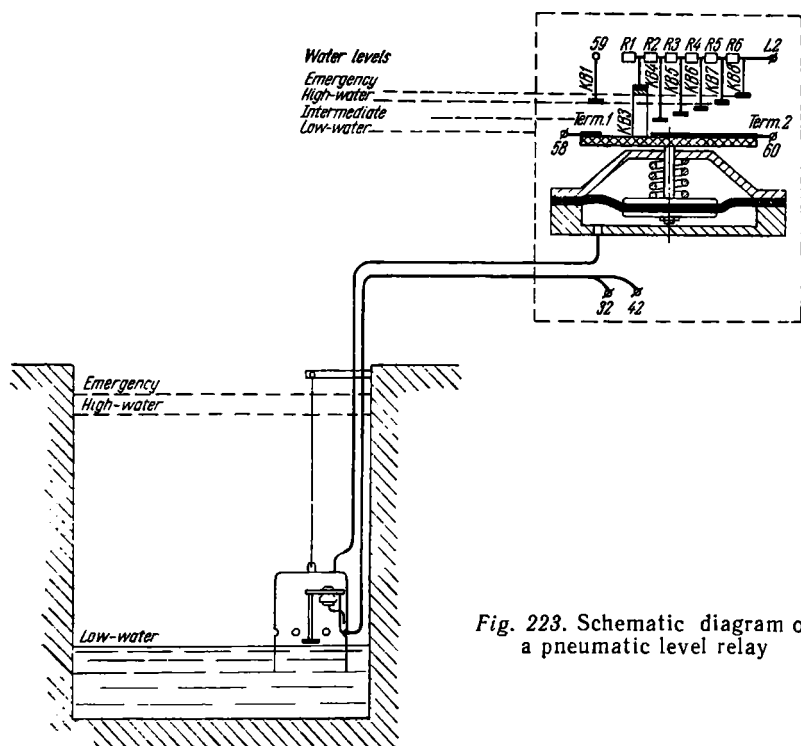
Fig. 223. Schematic diagram of a pneumatic level relay

The functions of the contact system and resistors $R1$ through $R6$ will be explained in the discussion of the ABH-1м (AVN-1m) circuit.

After some improvements, the ABH-1 (AVN-1) is now available as type ABH-1м (AVN-1m). Instead of the float-type relay, the ABH-1м (AVN-1m) circuit incorporates a more efficient electrode-type level transmitter or pickup and a more reliable РПФ-1 (RPF-1) flow-rate relay.

The electrode level pickup comprises an acid-resistant polyvinyl chloride tube in the lower part of which is fitted the electrodes which serve as the low-level contacts. The tube carries a movable ring mounting the high-level contacts. The ring can be set at the required height. At the top end the tube terminates in a box fitted with cable glands.

As was stated above, the operation of the pneumatic relay is duplicated (backed up) by a float relay, the construction of which can be seen in Fig. 224.
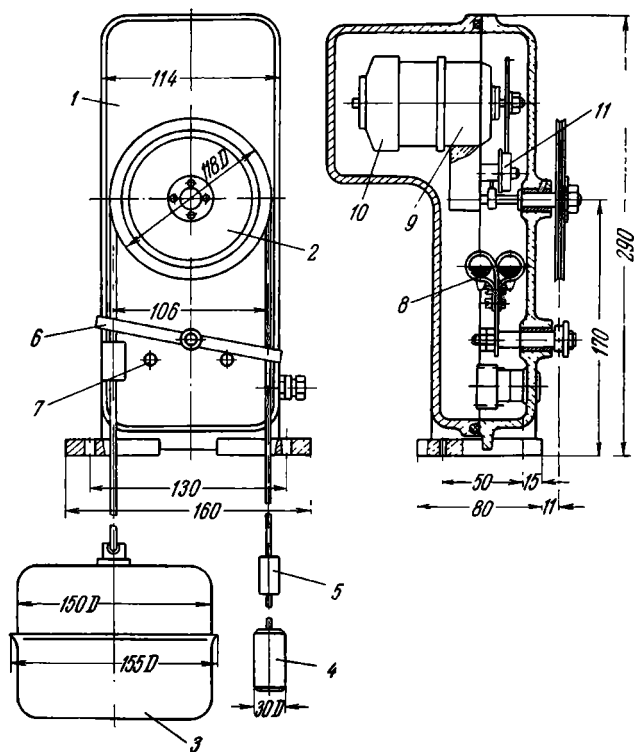
Fig. 224. Float level relay

On the outside of housing *1*, by means of a wire rope passing round sheave *2*, is hung float *3*, counterbalanced at the other end of the rope by counterweight *4*. As the water level in the sump changes, stops *5* attached to the wire rope turn switching lever *6* through an angle limited by lugs *7*. The shaft carrying lever *6* also carries a contact unit *8*. Float movement likewise causes sheave *2* to turn and rotate reduction gearing *9*. Through the latter, the sheave rotates a selsyn *10* (a special form of motor). The latter moves a remote selsyn receiver which indicates the level of the water in the sump. Change gears *11* permit the gear ratio to be adjusted to accord with the available travel of the float.

The constructional features of the РПФ-1 (RPF-1) flow-rate relay may be seen in Fig. 225. This relay is mounted on the pump suction
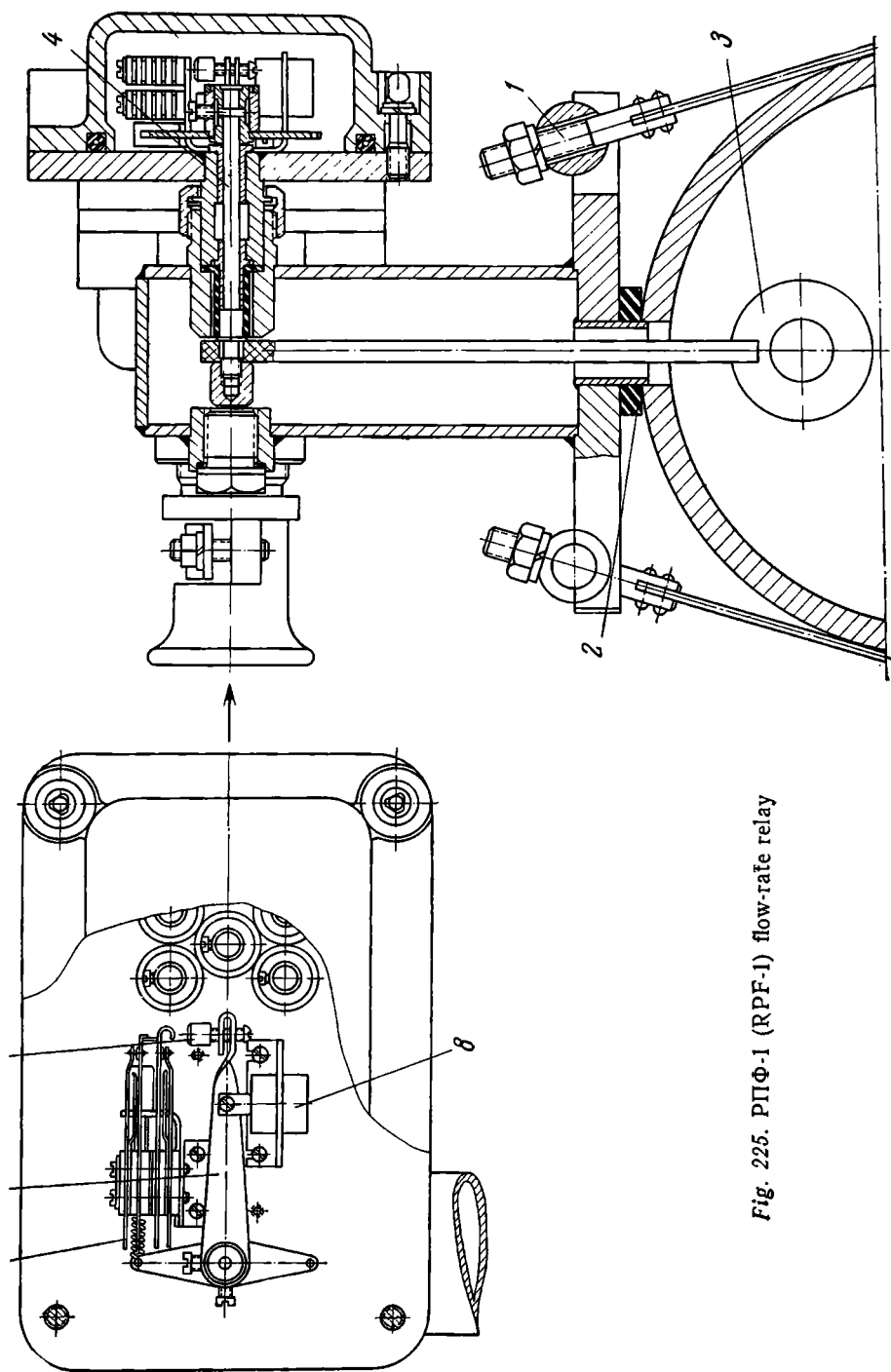
Fig. 225. РПФ-1 (RPF-1) flow-rate relay

line with a strap clamp *1*. The joint between the relay flange and the pipe is sealed with a rubber gasket *2*. When water flows in the line, it turns a flag *3* placed in the pipe. Rotation of the flag is transmitted through shaft *4* to lever *5*. The force on the lever is counterbalanced by a controlling spring *6*. As the lever turns, the pusher *7* causes the contact assembly to switch its normally-open and normally-closed contacts. To damp any vibrations of the flag due to turbulence in the water stream, the relay incorporates an oil dash-pot *8*.

Fig. 226 gives the elementary circuit diagram of the ABH-1м (AVN-1m) apparatus.

Normal Operation of the Circuit. After application of the supply voltage, the following occurs:

(1) Undervoltage relay *UV* picks up, opens its normally-closed contacts to cut off power to the motor of timing relay *MTR*. Relay *UV* is energized through the normally-closed contact *1DC1* of the timing relay;

(2) Signal relay *SR* picks up, closes its normally-open contact and switches on signal lamp *4SL* which shows that the voltage has been applied to the control apparatus and the illuminated indicator.

When the water level rises to the high datum, the contact *PRC1* of the pneumatic level relay closes to energize the motor of timing relay *2MTR*. The motor actuates the relay contact discs (see the lower right-hand side of Fig. 226) on which sets of contacts are fitted.

At first contact *1DC1* breaks and contact *LDC8* makes. Opening of the first contact does not de-energize relay *UV* because it remains connected to the supply through its normally-open contact *UV* (closed because relay *UV* has picked up) and through the closed low level contact (electrode *LLC*). When contact *1DC8* closes, it shunts the high level pneumatic relay contact *PR1*. Due to this, an unforeseen drop in water level and, consequently, opening of contact *PR1* will not interrupt the *2MTR* timing relay motor circuit.

As rotation of the timing-relay discs continues, contact *1DC2* closes the circuit of solenoid *1VS1* actuating the valve* which initiates priming of the pumps. Closing of contact *1DC4* establishes the circuit of the coil of auxiliary relay *1SAR* in the No. 1-pump magnetic starter box. Further rotation of the discs closes contact *1DC5*, energizes the auxiliary relay *1SAR* of the magnetic starter, and starts up pump No. 1. After this, contact *1DC2* opens to stop further priming of the pump.

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* A solenoid valve makes it possible to prime the pumps from a pressure or standpipe line.

At the end of the starting period, contact 1*DC7* is opened. This stops the 2*MTR* timing relay motor and closes contact 1*DC6* to energize hydraulic protection relay 1*HPR*. If the pump has developed a sufficient flow-rate by this moment, the flag-type flow-rate relay will close its 1*FR* contact in the *PR* relay coil circuit (see the lower left-hand part of Fig. 226) before contact 1*DC6* closes. This makes relay *PR* pick up to open its normally-closed contact in the 1*HPR* hydraulic protection relay coil circuit. At this moment the first pump is started up.

When the water level in the sump drops far enough, the lower level contact *LLC* of the electrode level detector opens, relay *UV* de-energizes, and normally-open contact *UV* interrupts the supply circuit to the priming control devices. Furthermore, the second normally-open contact *UV* interrupts the self holding circuit, while normally-closed contact *UV* completes the supply circuit to the 2*MTR* timing relay motor. The timing relay therefore becomes operative and brings its contact discs back to their initial position. This also shuts down the pump by opening the starter control circuit.

Hydraulic Protection. Protective relay *PR* is fed through a circuit containing the 1*FR* flow-rate relay normally-open contact and the contacts of temperature pickups 1*TP1*, 1*TP2*, 1*TP3* and 1*TP4*. The latter are embedded in the bearing housings of the associated pump unit and provide a check on their temperatures. Any excessive rise in temperature in the bearing makes the brass bulb detector increase in length and thereby separate a set of contacts. Thus, any noticeable drop in the pumping rate or overheating in the pump bearings causes relay *PR* to drop out. The relay then recloses its normally-closed *PR* contact to energize the hydraulic protection relay 1*HPR* (2*HPR*, 3*HPR*) and make it do the following switching in the circuit:

(a) its first normally-closed 1*HPR* contact opens to de-energize the No. 1 starter auxiliary relay 1*SAR* and shut down pump No. 1;

(b) its second normally-closed contact 1*HPR*, opens to interrupt the supply circuit to the pump priming control solenoid 1*VS1* (irrespective of the fact that it has already been disconnected by contact 1*DC2*) and thereby prevents any possible priming of a defective pump unit;

(c) its third normally-closed contact 1*HPR* opens to de-energize signal relay *SR* which closes its *SR* contact to complete the circuit to the siren and give an audible signal;

(d) its normally-open contact 1*HPR* closes to energize the 1*SL* (2*SL*, 3*SL*) signal lamp and 1*MTR* timing relay circuits to light up the visible alarm indicators;

(e) its normally-open contacts close to shunt the *1DC7* disc contact and start up the next pump if the water in the sump has reached its high level.

Undervoltage Protection. When the power supply is switched off, undervoltage relay *UV* is de-energized, drops out, and opens its self-holding normally-open contact. Reappearance of the supply voltage will therefore not lead to an automatic restart of a pump. Only after the timing relay contact discs are returned to their initial position will it be possible to start up a pump. These discs are turned by the *2MTR* timing relay motor the circuit of which contains the normally-closed contacts of relay *UV*.

Signalling. The ABH-1м (AVN-1m) circuit and apparatus will provide the following signals:

(a) a visible "normal-operation" signal with lamp *4SL* when the supply to the illuminated indicator and control apparatus is completed, the continuity of the signal circuits is not disturbed, the pumps are in order, and the water level in the sump is normal;

(b) an audible alarm signal when the supply to the control apparatus is interrupted, there is a break in the signalling circuit, an abnormal condition has developed in a pump unit, or the water level in the sump has reached a dangerous datum. This alarm signalling is effected by signal relay *SR*. An abnormality developing in the pump installation makes relay *SR* drop out and thus energize the audible alarm. To stop the alarm siren, signal cut-out button *SCB* is pushed to pick up alarm interlock relay *AIR* which opens its normally-closed contact and breaks the siren supply circuit;

(c) a visible signal when the pumps are in normal operation. It is initiated by the flow-rate relay which closes its contacts in the respective signal-lamp circuit;

(d) a blinking visible alarm signal when an abnormality develops in a pump. The signal is initiated by protective relay *PR* which drops out to energize the respective *HPR* hydraulic protective relay; the latter closes its normally-open contacts in the motor circuit of timing relay *1MTR* and the respective signal lamp circuit. The motor rotates at the rate of one revolution per second, and contact *1* of relay *1MTR* in the circuit of signal lamps *SL1*, *SL2* and *SL3* makes and breaks once every half-second to produce a blinking signal;

(e) indication of water level in the sump. The indicator used for this purpose is a moving-coil indicating instrument suitably graduated for direct reading. This instrument is connected in series with five resistors *R2* through *R6*, which are shunted out by respective pneumatic relay contacts *PRC4* through *PRC8* as the water level changes. The position of the pointer thus corresponds to the level of the water in the sump. The level indicator circuit also contains an emergency electrode pickup *EP* placed in the sump at the

dangerous water level. This pickup sends out a signal of dangerous water level in the event of a pneumatic relay failure.

The testing of a pump after a repair, or simultaneous operation of two or three pumps in the case of a dangerous rise in sump water level requires a change-over to manual control. This can be accomplished by a set of manual-control selector switches 1SS, 2SS, 3SS. When one of these switches is closed, the respective starter can be controlled by its push button station.

8-9. Remote Control of Conveyor Installations

The remote control of conveyors can be accomplished by either arranging a set of individual push button stations on a central control panel, or having one push button station to initiate an automatic start-up sequence for the entire conveyor line. In the first case, all the "Start" buttons have to be pressed one after another. In the second case, it is only necessary to press the button of the first conveyor. The other conveyors will then start up automatically, i.e., do not require any further action by the operator. Control in both cases is accomplished by one man from the central operator's station.
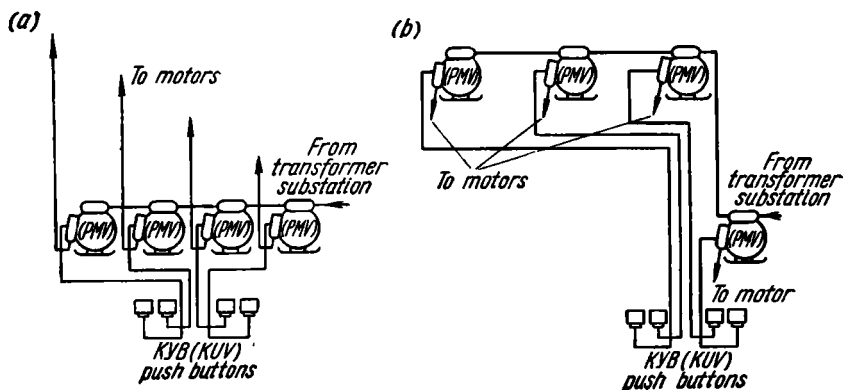
Fig. 227. Power distribution to conveyor motors:
a—radial; *b*—main-bus

Power can be fed to the conveyor motors by a radial or a main-bus circuit. In the radial scheme (Fig. 227*a*), a separate cable is run to each motor from its starter box, with all the boxes stationed at a distribution centre. A main-bus circuit (Fig. 227*b*) supplies all the conveyors through a single cable run from box to box stationed at their respective drive motors. The latter circuit finds application

in gate roads and inclines. The main-bus scheme is seldom used at headings because the magnetic starter boxes are not suitable for use at the faces.

Belt Conveyor Remote Control

Remote control is used for belt haulage systems installed in main workings (roadways, inclines, gates, cross entries). The conveyor control circuit performs the following operations:

(1) Automatically starts up the conveyor units consecutively and only after each preceding conveyor has come up to speed;

(2) Maintains an adjustable time delay between conveyor starts to avoid add-up of the motor starting currents;

(3) Provides a remote check on operation of each conveyor at starting and shows the number of conveyors operating in the line;

(4) Automatically stops the motor of a defective conveyor and all other conveyors carrying coal in its direction and onto it;

(5) Starts up any desired number of conveyors in the line;

(6) Independently starts up any motor in the line for inspection, repair, etc.;

(7) Provides two-way signalling and warning prior to starting.

Fig. 228 gives the layout diagram of an automatic line consisting of three conveyors driven by three motors M_1, M_2 and M_3.

Remote control of the conveyor line is effected by hydraulic speed relays HSR, control panel CP, relay boxes RB, magnetic starter boxes ПМВ (PMV), sirens S, and push button stations PB.

A hydraulic speed relay comprises a rotary-type oil pump, time delay device and a contact closing mechanism. The time delay device is a cylinder containing a suitably shaped piston held in its lowermost position by a return spring. To limit piston travel and adjust the time delay setting of the relay, a special adjusting screw is provided.

When the pump rotor is put into motion, oil from the relay body is delivered to the time delay device cylinder and brings the piston up against the setting stop screw. The time required for this upward travel determines the closing time delay. After filling the time-delay cylinder, the oil enters the closing mechanism cylinder to force its piston upward and thereby close the control circuit contacts.

Control panel CP serves to start and stop the conveyor line, sound a warning signal and indicate the number of conveyors in operation at any given moment of time. The control panel is stationed at the delivery end, near the drive head of the first conveyor. The front panel of the control panel accommodates four push buttons: "Start", "Stop", "Cut-out" and "Signal". By means of the first two buttons the line is started or stopped. The "Cut-out" button
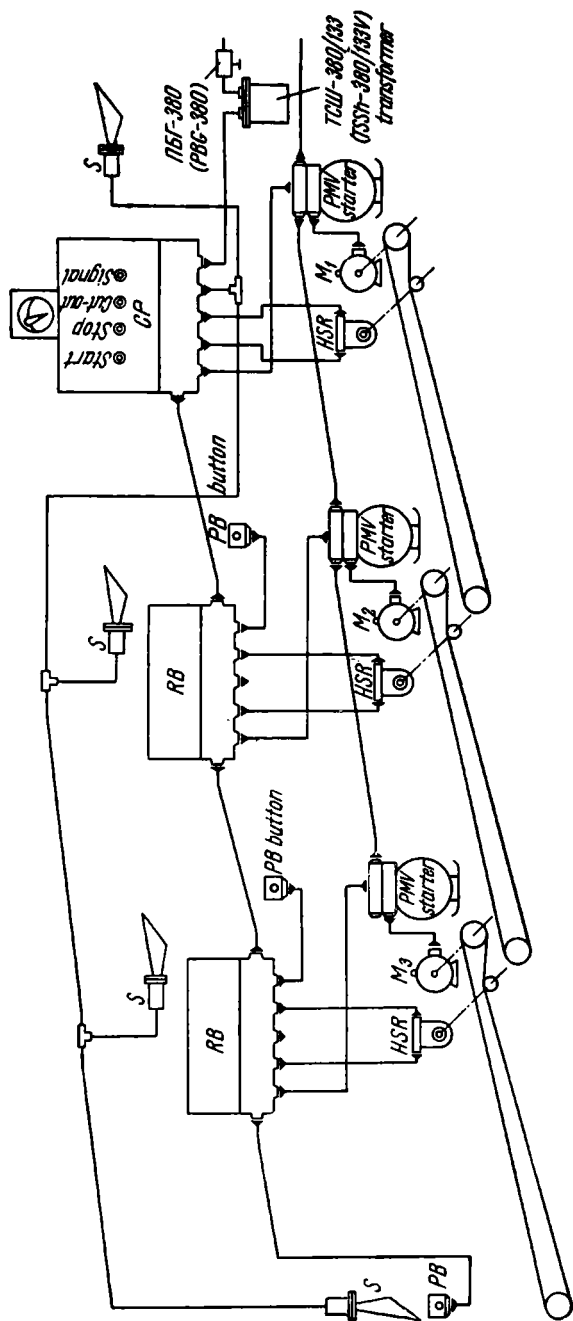
Fig. 228. Schematic representation of an automatically controlled line of conveyors

makes it possible to limit the number of conveyors put into ope-
ration. For example, if it is necessary to operate the first two con-
veyors, the "Start" button is pressed to start up the line and the
indicator pointer is watched. As soon as its pointer reaches "2" on
the scale, thus indicating that the second conveyor has begun ope-
ration, the "Cut-out" button is pressed to prevent starting of the fol-
lowing conveyors, while leaving the first and second conveyors in
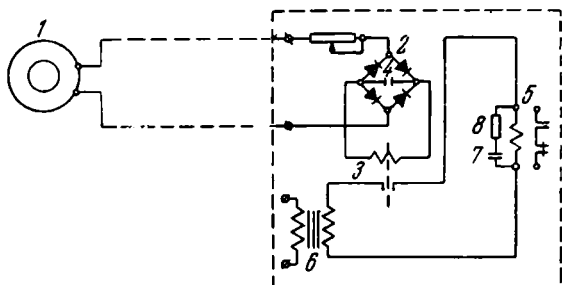
Fig. 229. Circuit of ВИРС-2 (VIRS-2) electric speed relay

operation. The "Signal" button serves to energize the warning
sirens. The latter can also be operated by any one of the *PB* button
stations provided at the drive unit of each conveyor.

The indicator showing how many conveyors are in operation is
mounted on the top wall of the control console. This indicator is
a moving-coil instrument operating from the secondary winding
of a current transformer through a copper oxide rectifier. The cur-
rent transformer is connected so that the starting of every further
conveyor raises the current passing through its primary, and a
greater emf is induced in the secondary winding. By reason of this,
the start-up of the first conveyor makes the indicator pointer move
to scale division 1; the start-up of the second conveyor, to scale
division 2, and so on.

Relay boxes *RB* serve as enclosures for circuit components and
also as cable boxes which connect the separate units of the system
into an integral circuit. The relay boxes are set up, one at the drive
head of each conveyor, starting with the second.

Instead of hydraulic speed relays, automatic conveyor lines now
use ВИРС-2 (VIRS-2) electric speed relays. The circuit diagram of
such a relay can be seen in Fig. 229. For speed pickup, a miniature
a.c. generator *1* is coupled to the shaft of one of the rollers supporting
the return belt strand. When the conveyor operates, the generator
is maintained in rotation and sends a current through rectifier *2*
to energize relay *3*. Capacitor *4* functions to smooth out the pulsa-

tions of the rectified current. Through the closed contact of relay *3*, current is fed to control relay *5* from transformer *6*. The control relay contacts (one normally-open, the other normally-closed) are inserted in the conveyor line control circuit. In the event of a shut-down of the conveyor or breaking of its belt, the generator comes to a stop and thus de-energizes control relay *5*. Resistor *8* and capacitor *7* are included to reduce arcing and prolong the life of the relay contacts.

Remote Control of Scraper-chain Conveyors

Remote control circuits for scraper-chain conveyors provide:

(1) Interlocking between starter boxes so that the conveyors can be started up according to a fixed sequence;

(2) Automatic stopping of a damaged conveyor and all other con-veyors feeding coal in its direction when the damage involves a stoppage of the chain (for example, when the chain breaks);

(3) Local start-up of any individual conveyor of the line for test-ing or adjustment;

(4) Warning when the line is put into operation.

An automatic check on the condition and movement of the chain is now provided by non-contact РУК (RUK) apparatus comprising a set of induction pickups, and a control unit. The induction pickup consists of a casing containing a set of coils within which permanent magnets are placed. A pickup for single-chain conveyors has two series-connected coils (Fig. 230) and is installed under the return strand of the scraper chain on the underpan of the first conveyor-section. The scraper chain thus travels above the pickup. On a double-chain conveyor the pickups are placed above each of the top chain strands at the first upper trough. The pickup casing is provided with a plug cable-entrance fitting for flexible cable.

The control unit is housed in a flameproof enclosure and consists of a semiconductor amplifier, voltage stabilizer, speed relay and an auxiliary relay. It is fed at 36 V from the starter box step-down transformer and is connected in parallel with the auxiliary relay [in ПMB-1331 (PMV-1331) boxes] or the contactor coil [in ПMB-1344 (PMV-1344) boxes].

As the scraper chain travels, the magnetic flux of the permanent magnets is periodically "short-circuited" through its links and thus induces an emf in the pickup coils. This emf is amplified by the semiconductor amplifier and is applied to the relay, the contact of which is connected in the control circuit of the magnetic starter.

When the conveyor operates normally, this contact remains closed. But, if the scraper chain stops (on a single-chain conveyor),

or a broken portion of the chain goes past the pickup (on a double-chain conveyor), no emf will be induced in the pickup coils, and the contact opens to cause the magnetic starter to drop open.

As distinct from the РУК (RUK) system, the ДДЦ-1 (DDTs-1) induction pickup responds to travel of the scraper flights and not
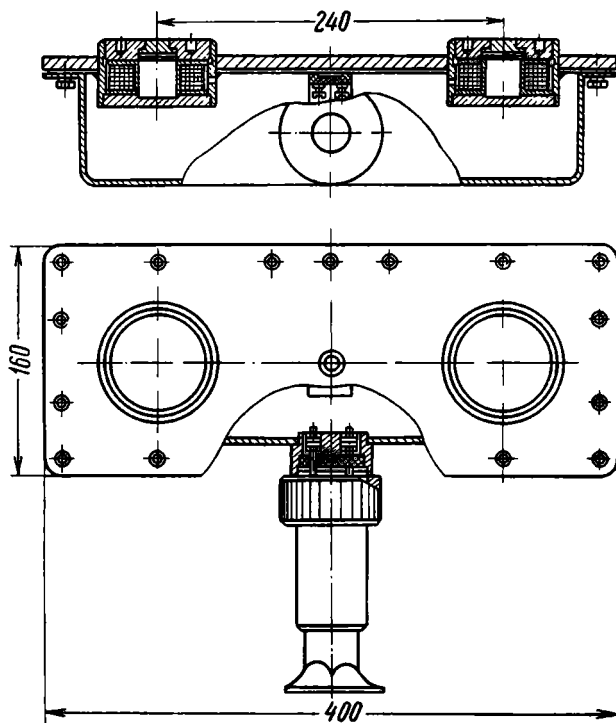
Fig. 230. Features of an induction speed pickup

the chain links. The pickup signals are received by a sensitive moving-coil type relay. In the ВИРС-2с (VIRS-2s) apparatus the pickups, as in the РУК (RUK) apparatus, respond to chain-link travel, and the signals are received by a sensitive polarized relay. Fig. 231 illustrates the layout of automatic control of scraper-chain conveyors operated from a main-bus cable circuit. Fig. 232 gives the diagram of the control circuit using ПМВ-1331 (PMV-1331) magnetic starter boxes.

The magnetic starter boxes *1* (Fig. 231) located at the conveyor drives are interconnected by a six-conductor flexible power cable. The РУК (RUK) control units *2* are each connected to its associated

box and induction pickup *3*. Control of the conveyor line is carried
out with a two-button station *4* placed at the head-end conveyor.

The line is started up by pressing the "Start" button and holding
it down until the entire line begins to operate. This button completes
the circuit of relay R_1 (Fig. 232) from transformer Tr_1 of the first
starter, to "Start" button, germanium rectifier (diode) $D1_1$, relay
coil R_1, diode $D4_1$ and back to transformer Tr_1. Relay R_1 operates
to close its contact in the circuit to starter auxiliary relay AR_1,
the latter closes its AR_1 contact and thus energizes magnetic starter
operating coil $K1$. At the same time power is supplied to the stab-
ilizer ST_1 of the first РУК (RUK) control unit.*
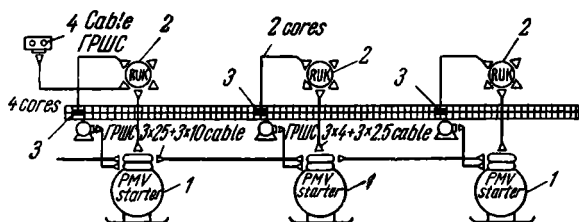
Fig. 231. Arrangement of equipment for remote con-
trol of scraper-chain conveyors with main-bus scheme of
motor supply

When the scraper chain comes up to normal speed, the pickup
energizes speed relay SR_1 which closes its contact in the circuit
to relay R_2 at the second conveyor (relay R_2 is energized by the
current flowing from transformer Tr_1, to "Start" button, contact
SR_1, diode $D1_2$, relay coil R_2, diode $D4_2$ and back to transformer
Tr_1). Relay R_2 then energizes auxiliary relay AR_2, and the latter
closes the second-conveyor starter. Start-up of the remaining con-
veyors then follows in the same manner.

As soon as the last conveyor sets into motion, the "Start" button
may be released. Relays R_1, R_2, etc., are held in by the current
flowing through the respective speed relay contacts as follows:

(a) *for relay R_1:* from lower terminal of transformer Tr_1, through
diode $D3_1$, relay coil R_1, diode $D2_1$, speed relay contact SR_1,
relay contact R_1, diode $D5_1$, "Stop" button, and back to upper
terminal of transformer Tr_1;

(b) *for relay R_2:* from lower terminal of transformer Tr_1, through
control conductor in power main cable, diode $D3_2$, relay coil R_2,

* The stabilizers in the circuit function to maintain a constant voltage across
the РУК (RUK) units, since the voltage fluctuates greatly, especially when the
conveyor-line motors are started.

Fig. 232. Remote control circuit of scraper-chain conveyors with main-bus scheme of power supply

Symbols:

● Through-stud contact
  in PYK(RUK) control box

◎ Auxiliary through-stud contact
  in starter contactor

⦿ Through-stud contact
  in push button station

diode $D2_2$, speed relay contact SR_2, control conductor in power main cable, speed relay contact SR_1, relay contact R_1, diode $D5_1$, "Stop" button, and back to upper terminal of transformer Tr_1.

If the scraper chain stops or breaks, the respective speed relay contact opens to shutdown the damaged conveyor and all the conveyors carrying coal in its direction. The other side of the conveyor

Fig. 233. Arrangement of equipment for remote control of scraper-chain conveyors with radial scheme of power supply

line remains in operation. After the trouble has been remedied, the stopped part of the line can be restarted by pressure on the "Start" button at the repaired conveyor or the "Start" button at the loading point.

To bring the conveyor line to a stop, the control station at the loading point includes a "Stop" button. Local starting or stopping of any conveyor in the line can be accomplished with the push buttons of the respective starter box. Complete start-up of the line is signalled by lighting up of a lamp at the loading point. This indicating lamp is switched on by the contact of the tail-end conveyor relay R (both omitted in the circuit diagram).

Signalling circuits also employ the РЭУВ (REUV) intrinsically-safe apparatus which utilizes bare wires. The arrangement of the equipment for radial-circuit supply of a conveyor line is shown in Fig. 233; the control circuit of such an installation is given in Fig. 234. In this layout the magnetic starter boxes and the РУК (RUK) control units are stationed at the distribution centre near the loading point of the face gate end. Each of the conveyor line motors is connected by a separate six-conductor cable to the respective starter box *1* and by control cables to the induction pickups *3*.

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The control units *2* are directly connected to the starter boxes. At the motors the cables are terminated in junction boxes *4*. The line is controlled by push buttons, an ordinary two-button station *5* being used for the first conveyor, and special type КПУ-3 (KPU-3) push button stations *6* for the other conveyors. A КПУ-3 (KPU-3) station (Fig. 235) has three push buttons: "Start", "Stop" and "Cut-out". A lever interlinking the "Start" and "Cut-out" buttons prevents them from being closed simultaneously. The station casing also incorporates a built-in switchboard-type indicating lamp.

To start the first conveyor, the "Start$_1$" button (Fig. 234) is pressed so that relay R_1 is energized by a current which flows from upper terminal of transformer Tr_1 in the first magnetic starter, through "Start$_1$" button, diode $D1_1$, relay coil R_1, diode $D4_1$, and back to lower terminal of transformer Tr_1. Relay R_1 closes its contact in the circuit of starter auxiliary relay AR_1 and thereby closes the first-conveyor starter. Power is at the same time supplied to the voltage stabilizer CT_1 of the РУК (RUK) unit. When the chain comes up to normal speed, the relay SR_1 of the first РУК (RUK) unit operates to close its SR_1 contact, and the "Start$_1$" button may be released. Now current flows from lower terminal of transformer Tr_1, through diode $D3_1$, relay coil R_1, diode $D2_1$, speed relay contact SR_1, relay contact R_1, diode $D5_1$, "Stop$_1$" button, and back to upper terminal of transformer Tr_1.

Fig. 235. Circuit of КПУ-3 (KPU-3) push button station

Start-up of the second conveyor can only be carried out after the first conveyor has been put into operation because the circuit of relay R_2 contains the normally-open contact of relay R_1. Starting is accomplished with the "Start$_2$" button of the КПУ-3 (KPU-3) station. Its contacts complete the circuit of relay coil R_2 from upper terminal of transformer Tr_2, through relay contact R_1, "Start$_2$" button, diode $D1_2$, relay coil R_2, diode $D4_2$, and back to lower terminal of transformer Tr_2. Signal lamp SL_2 lights up simultaneously by current flowing from upper terminal of transformer Tr_2, through signal lamp SL_2, starter contactor interlock CI_1, "Stop" button, and back to lower terminal of transformer Tr_2. Start-up of the other conveyors takes place in the same manner. Button "Stop," at the first station is pressed when it is necessary to shut-down the entire line. When a scraper chain stops or breaks on any one of the conveyors, it results in opening of the respective speed relay contact (SR) to stop the damaged conveyor and all the other

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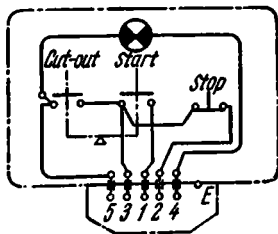
conveyors passing coal towards it. All the signal lamps go out, starting at the push button station of the damaged conveyor.

The starting and stopping of each conveyor directly at its drive unit is done with a standard two-button station having a "Start" and a "Stop" button. This push button station makes it possible to give the conveyor an individual trial run after a repair or set-up at a new site. For this, the "Cut-out" button on the respective КПУ-3 (KPU-3) station (placed at the delivery end) must be pressed and locked in. Due to the mechanical interlock between the "Start" and "Cut-out" buttons the given conveyor cannot be started from the delivery end.

Chapter IX

MINE CABLE CIRCUITS

9-1. Types of Cables and Methods of Installation

Power transmission and distribution in coal mines may be carried out only with the use of armoured cables for stationary equipment and trailing cables for mobile face machinery.

An armoured power cable consists of three copper conductors, or cores, insulated with impregnated paper tape. The three insulated conductors are twisted together into a cable and again impregnated paper tape is applied over all to provide the belt insulation. Over this belt insulation a continuous lead sheath is applied to protect the cable from moisture. As protection against corrosion, a covering of impregnated jute yarn is applied over the lead sheath. After this, the cable is armoured with steel tape or wires to provide mechanical protection, and the armour is served with an outer covering of impregnated jute to hinder attack of the armour by rust.

The conductors of a trailing cable are made up of a large number of tinned wires and are each insulated with two layers of rubber and a rubberized cotton braid.

The insulated conductors are laid up into a cable which is then covered by two layers of tough rubber between which a serving of rubberized tape is provided. These cables have an earth core with a covering coloured to distinguish it from the power cores.

Three-conductor, lead-sheathed cables armoured with steel tape are used for stationary installation in horizontal workings, while shafts and inclined workings require drained-type steel-wire armoured cables (for shafts deeper than 300 metres).*

Cables should be laid on flexible type hangers in timber- and metal-supported workings; at the faces, on wooden pegs, and along

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* Drained-type cables contain a minimum of impregnating compound as a precaution against leakage of compound from the lower sealing end.

gate roadways, on wooden brackets or hung from canvas belt loops. Cables should be laid with a small sag between supports to avoid excessive tension in them.

Cable runs within the substation and switchgear chambers must not have an outer jute covering, and the cable armour should be given a coat of varnish for protection against corrosion. In roadways where coal haulage is by trolley locomotives, the cables must have impregnated jute coverings to prevent corrosion by stray currents. The cables should never be enclosed in wooden troughing.

Supports for steel-tape armoured cables should be spaced not more than three metres apart.

Rigid clamping of armoured cables may be used in concrete and brickwork faced workings and also in workings in strong rock not requiring support.

Trailing cables, if they remain connected to starter boxes, must not be coiled in single-loop or figure-of-eight fashion. They should be laid out in continuous runs hung from supports provided along the working.

Paper-insulated armoured cables require thorough jointing and termination in compound-filled types of boxes and sealing-ends seals.*

Trailing cables have to operate in very adverse conditions and therefore require regular checking. They must be inspected at the beginning of each shift for the continuity of the rubber insulation by the operator of the unit supplied by the cable. A damaged cable must immediately be repaired.

Since the quality of such a repair is vital for the safety of personnel, trailing cables must always be vulcanized after a repair. Vulcanization should be done in a surface or underground cable repair shop. Cables with unvulcanized joints should never be used.

Trailing cables of coalcutters, cutter-loaders, or hand or post-mounted drills and having more than four vulcanized joints within any 100 metres of length should be rejected.

If shot firing is to be done near a trailing cable, the cable must first be moved to a safe place or laid on the floor under the conveyor troughs or protected with boards.

High-voltage armoured cables require correct jointing. The end of the cable remaining on a reel after the requisite length has been run out and cut off for installation should be sealed with a lead cap burned over the end of the sheath to shut out moisture. A temporary seal may consist of a layer of hot compound covered with tape and another layer of compound on top.

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* Armoured cables for up to 1000V may be terminated by a "dry" and sealing method not requiring a filling box.

Before an armoured cable is installed, a piece 50 cm long is cut from its end to check the insulation for the presence of moisture. The check is performed by dipping samples of the conductor, belt and interstice paper insulation into molten paraffin or cable compound. The presence of moisture will be indicated by bubbles rising to the surface with a light crackling sound.

Before lengths of cable are jointed, the cable cores must be tested for continuity and the insulation resistance measured relative to each other and to the lead sheath.

9-2. Cable Fault Location

Faults in cables are located with a megohmmeter. A break in a cable conductor (see Fig. 236) is found as follows: both ends of the conductor, for example *1* and *1'*, are connected to the terminals *4-4'* of the megohmmeter. If the conductor is continuous, a current
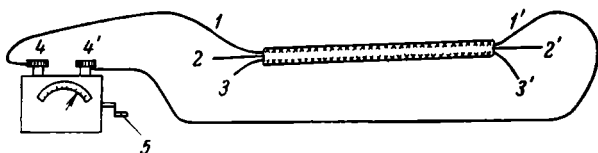
Fig. 236. Open-circuit test of a cable conductor

will flow from terminal *4* through cable run *1-1'* and back to terminal *4'* when the megohmmeter handle *5* is cranked, and its pointer will deflect. When the pointer fails to move, it shows that the circuit is broken. Conductors *2-2'* and *3-3'* are tested identically.

Conductor continuity can be tested in another manner, especially when the cable is long. The conductors are connected together at the far end (*1'* and *2'*) and connected in pairs (*1* and *2*) to megohmmeter terminals *4-4'* at the test end. The path for current will be from terminal *4* through cable *1-1'* and *2-2'* back to terminal *4'*. Cores *2* and *3* are checked similarly.

A short in a cable is located by connecting the megohmmeter terminals in turn to two cable conductors while the other ends are isolated from each other (Fig. 237). If there is no short, the megohmmeter pointer will not deflect when the handle is cranked. Should a short exist, the pointer will deflect full-scale.

The megohmmeter also serves to test the insulation of a cable or unit of electrical equipment. For this, one of its terminals is connected to the winding of a motor or cable core, while the other terminal is connected to the motor frame or cable armour. If the

insulation is good, the megohmmeter pointer will read either zero or the insulation resistance of the winding or cable core to earth.

Faults in armoured and trailing cables in gas and coal-dust hazardous mines can be located by sending an alternating current of audio frequency through the defective conductor. This is done by means of the ИПК-2 (IPK-2) tester. The tester accurately pin-points the fault, and repairs may be made with the least loss of time. The tester, however, will not locate an "open" (i. e., a break) in a cable.
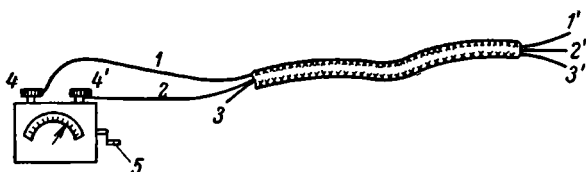
Fig. 237. Short-circuit test of cable conductors

The tester consists of an audio-frequency valve signal generator, an amplifier, headphones, and a searching coil. The generator is housed in an aluminium flameproof casing accommodating the necessary batteries for generator supply.

The amplifier (receiver) is a three-valve unit in a flameproof case which also contains its power supplies. The searching coil is encased in a moulded-plastic body having a handle. The amplifier is carried on a shoulder strap.

The procedure for locating shorts or earth faults is as follows:

1. The faulty cable is disconnected at both ends.

2. The armour jumpers are checked for breaks or broken connections at the joints.

3. The fault is identified with a megohmmeter to know whether it is a single, double or triple core fault to the armour, or a core-to-core short due to defective insulation.

4. The lead sheath and the armour are cleaned bright and earthed to the general mine earthing system at the end where the tester generator is to be connected.

5. The damaged elements of the cable are connected to the generator terminals (two cores or a core and the lead sheath) and the generator plug is inserted in its supply socket.

6. The earphone and searching-coil jacks are plugged into their T (telephone) and R (receiver) sockets on the amplifier.

7. The amplifier is now slung over the shoulder, and both the generator and amplifier are switched on. To be sure that the tester operates normally the searching coil is brought up to the generator case. A strong characteristic note (850-900 c/s) should be heard in the earphones.

8. The searching coil is placed on the cable and moved along so that its edge touches the armour.

As the coil is moved along, the characteristic note of the generator heard in the earphones will vary in volume in a periodic manner, reaching a maximum at some points and a minimum at others. The note will cease to vary when the fault has been passed.

9-3. Trailing Cables

The mining cables used in coal mines in the Soviet Union are of the following types:

(a) ГРШС (GRShS)—rubber-sheathed cable with rubber-insulated conductors laid around a cradle centre (production discontinued in 1961);

(b) ГРШ (GRSh)—rubber-sheathed similar to that above, but without a cradle centre;

(c) ГРШСНЭ (GRShSNE)—non-burning, rubber-sheathed, rubber-insulated and screened cable.

The above cables may be of four-, five-, or six-core construction. Table 14 shows the number of cores in cables of different size, while Table 15 gives the sizes of the earth and pilot cores for the respective power core sizes.

<table>
<tr><td colspan="4" align="center">Table 14</td><td colspan="2" align="center">Table 15</td></tr>
<tr><td rowspan="2">Power core size. mm²</td><td colspan="3" align="center">Number of cores</td><td colspan="2" align="center">Core size, mm²</td></tr>
<tr><td>Power</td><td>Earth</td><td>Pilot</td><td>Power</td><td>Earth or pilot</td></tr>
<tr><td>2.5</td><td>3</td><td>1</td><td>1</td><td>2.5</td><td>1.5</td></tr>
<tr><td>2.5-70</td><td>3</td><td>1</td><td>—</td><td>4</td><td>2.5</td></tr>
<tr><td>4-50</td><td>3</td><td>1</td><td>2</td><td>6</td><td>4</td></tr>
<tr><td></td><td></td><td></td><td></td><td>10</td><td>6</td></tr>
<tr><td></td><td></td><td></td><td></td><td>16-70</td><td>10</td></tr>
</table>

For example, a cable of 35 mm² size may have either four or five cores with an earth and a pilot core of 10 mm² size in both cases. The latter cables are hence designated, respectively, as: ГРШС $3 \times 35 + 1 \times 10$ and ГРШС $3 \times 35 + 3 \times 10$.

The mine electrician must keep constant watch on the trailing cables. They are the most vulnerable elements and a source of trouble in the district power supply system due to their extensive length, low strength of the sheaths, and exposure to adverse surroundings.

Most often trailing cables are damaged when dragged over the floor, struck by falling rock and coal, crushed by a prop, etc. As often, they may be cut by tools, sharp edges of coal or rock, and be injured by machines and shot-firing.

Cable damage may be classed as: damage to the sheath and damage to the inner insulation or cores.

In the first case the repair is made with a self-vulcanizing paste. In the second case, when the insulating layer, a broken core, or a joint has to be dealt with, more involved repair and vulcanizing of the damaged section is necessary.

Trailing cables should be brought up to the surface shop at quarterly intervals for a check-up. Any defects detected in the sheaths are then repaired. If a cable is injured or broken during a shift, it must immediately be taken out of service, repaired, and the faulty spot hot-vulcanized.

A faulty trailing cable is a source of great danger to the men because of the electric shock hazard, and may also cause a fire or explosion of fire damp. For greater safety, trailing cables are now made of the screened type.

The ГРШСНЭ (GRShSNE) cable, which is of this type, is shown in Fig. 238.

The power and pilot cores are rubber-insulated as usual, but the earth core has a covering of semiconductive rubber.* All the cores are laid round a rubber cradle centre and protected by a common semiconductive rubber covering (screen) over which the outer tough rubber sheath is applied. The common screen is electrically connected with the earth core; therefore, should the cable be damaged the screen will come into contact with a power core, causing the earth leakage relay to operate.

Key:

▣ *Copper cores*

☐ *PTH-1(RTI-1) insulating rubber*

▨ *Rubberised insulating tape*

■ *Semi-conductive rubber*

▦ *Tough rubber sheath*

▦ *Cradle of PTH-1(RTI-1) insulating rubber*

Fig. 238. Cross section of the ГРШСНЭ (GRShSNE) flexible screened cable

9-4. Armoured Cables

Main supply circuits in horizontal and low gradient workings (up to 45 degrees) use СБ (SB) and СБГ (SBG) armoured cables. СБ (SB) cable is lead-sheathed, paper-insulated, double-steel-tape armoured, with an outer covering of impregnated jute yarn. As distinct from

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* Semiconductive rubber has a high carbon black content to make it sufficiently conductive for electric current and provide reliable operation of the earth fault protection when a cable is damaged.

it, СБГ (SBG) cable has no yarn covering. The largest size of armoured cable used in coal mines is 95 mm², as the cable boxes of mining electrical apparatus will not take a bigger size.

Where the headings are inclined more than 45 degrees, flat-wire-armoured СП (SP) or СПГ (SPG) lead-sheathed, paper-insulated cables are installed. The former cable has, in addition, an impregnated jute covering over its flat-wire armour.

The steel wire armour of such cables protects the lead sheath from mechanical injury and takes the tensional stresses without transferring them to the power cores.

Vertical runs in shafts deeper than 300 metres are put in with wire-armoured, lead-sheathed СПНШ (SPNSh) cable having pre-dried paper insulation.

9-5. District Circuit Layout Diagrams

For visualization and proper use of power distribution in his area, the district engineer is duty-bound to keep an up-to-date circuit layout diagram showing all the cables including their grade, size, length, the associated switchgear, protective devices and loads. Such diagrams are drawn with conventional symbols as shown in Fig. 239.

The diagram should also indicate the two-phase short-circuit currents at the furthermost points in the circuits, the overcurrent relay settings, and the current ratings of the fuse links. Arrows on the diagram should show the direction of the intake air stream.

The diagram must be approved by the chief electrical engineer of the mine. As extensions and withdrawals are made, the diagram should be revised. It is drawn in two copies; one the master, the other the working copy. The master diagram is kept by the district mechanic, and the working diagram by the shift electrical fitter.

9-6. Voltage Drop in Cable Circuits. Selection of District and Inbye Substation Transformers. Cable Size Calculation

By Soviet standards, an electric motor must continue to operate normally at a supply voltage within 5 per cent of the rated voltage, (the value marked on the motor nameplate). Thus, a motor rated for 380V will operate normally with the supply voltage between the following limits:

$$380 + \left(\frac{5 \times 380}{100}\right) = 399 \text{ V} \quad \text{and} \quad 380 - \left(\frac{5 \times 380}{100}\right) = 361 \text{ V}.$$

Therefore, for the motor to operate normally, the total voltage drop in the circuit (the transformer secondary and the armoured

Key:

⌐ Donbass cutter loader
Ⅱ MK-3 haulage winch
▯ Two-button KУБ-6П2(КУВ-6П2) push button station
▯ Single-button КУБ-6П2(КУВ-6П2) push button station
○ Individual ПГ-60(RP-60) lighting fitting
⌐ Water-spray pump

ГРШС cable 3×4+1×2.5:200
ГРШС cable 3×35+3×10:200
ГРШС cable 3×16+2×4:200
ГРШС cable 3×4+1×2.5:100
ГРШС cable 3×25+1×10:1200
ГРШС cable 3×16+1×10:80

ГРШС cable 3×16+1×10:80

ГРШС cable 3×10+1×6:50

ГРШС cable 3×8+4:60

ГРШС cable 3×16+1×10:80

ГРШС cable 3×6+ +1×4.5

СБ 3×35:300

ГРШС cable 3×2.5+1×1.50
ГРШС cable 3×16+1×10:50

ГРШС cable 3×2.5+1×1.5:30

ГРШС cable 3×18+1×6.5   ГРШС 3×4×2.5:5

Fig. 239. Typical district power circuit layout diagram

and flexible cables) may be

$$400 - 361 = 39 \text{ V},$$

where 400 is the no-load voltage of the transformer secondary when rated voltage is applied to the high-voltage side.

If the rated voltage is 660V, the permissible voltage drop is

$$693 - 627 = 66 \text{ V*}.$$

At excessive voltage drops the efficiency of the machines falls off appreciably, their motors become unable to handle overloads, and even stall. The motors also draw larger currents at decreased voltages and therefore the windings may reach dangerous temperatures.

At starting, the voltage drop in the circuit is considerably increased because of the large starting current drawn by the motor.

The value of the voltage across the terminals of cutter-loader and coalcutter motors at starting may be allowed to drop to anywhere between 245 and 255V for a supply voltage of 380V.

In selecting the sizes of the cables for a mine circuit, the permissible voltage drop in the circuit must be taken into account. The procedure for calculating a cable circuit follows.

1. Draw up a diagram of the district circuit to be supplied from the given district substation. It should show the grade and length of all the cables, and the power ratings and currents of the associated loads. For cutter-loader and coalcutter motors, the 60-min rating is used; for the motors of other machines and tools, the continuous rating.

Table 16 gives the starting and running currents of commonly used 380-V motors, and also the overcurrent relay settings and the fuse link current ratings recommended for short-circuit protection.

2. Select the district substation transformer by adding together the current ratings of all the loads and taking a transformer capable of carrying the total current.

The rated current of a coalcutter or cutter-loader motor is considered to be its 60-min current rating; the motors of other units, the continuous-rating current.

3. Select the size of trailing cables from Table 17 for the rated currents of the motor. The table also gives the maximum current ratings of fuse links, the lowest two-phase short-circuit current, the least size of trailing and armoured cables and the maximum current-carrying capacities. Recommended practice is that the trailing cable for a coalcutter should be not less than 25 mm² in size; that for a cutter-loader, not less than 35 mm².

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* 693V is the no-load voltage of the transformer secondary and 627V is the lowest permissible voltage for the motor, as $627 = 660 - \left(\dfrac{5 \times 660}{100} \right)$.

Table 16

Machine or mechanism	Motor type designation	Nominal value			Overcurrent relay setting, A	Fuse link current rating, A
		Power, kW	Rated current, A	Starting current, A		
КМП-2 (KMP-2) coal-cutter	МА-191/10	47	96	510	500	200
Donbass cutter-loader:						
main motor	МАД-191/11k	65	131	519		
loader motor . . .	МАР-6-11/4	13	28	200	700	—
	МАД-191/35г	32	70	320		
Gornyak cutter-loader:						
main motor	МАД-191-11мг	65	131	519		
loader motor	МАД-191-35г	32	70	320	600	—
ППМ-4 (PPM-4) rock loader:						
main motor	КО-31-6	20	41	208		
conveyor motor . .	К-11-4	4	8.5	50	220	125
ЭПМ-1 (EPM-1) rock loader	КТСВ-110/755	2×10.5	2×22.8	22.8+137	160	80
ПК-2м (PK-2m) entry-type cutter-loader:						
main-drive motor .	МА-144-2/4	29	56.2	394		
conveyor motor . .	ВАД-27	2.7	6	42	400	200
crawler motor . . .	ТАГ-32/4	3.5	7.4	52	—	—
СТ-6 (ST-6) conveyor	ТАГ-41/4	6.3	13.1	92	100	60
КС-10 (KS-10) conveyor	КОФ-12-4К	11	22.5	112	120	60
СКР-11 (SKR-11) conveyor	МА-143-1/4	11.4	23	138	140	60
СКР-20 (SKR-20) conveyor	КО-22-4	20	40	240	250	125
СТР-30 (STR-30) conveyor	МА-144-2/4	29	56.2	394	400	200
	КО-32-4	32	63	378	400	200
КС-9 (KS-9) conveyor	КОФ-32-4	32	63	378	400	200
КРШ-220 (KRSh-220) conveyor	МА-146-1/8	35	70	420	450	200
РТУ-30 (RTU-30) conveyor	КО-22-4	21.5	28	138	140	80
МЭЛ-4.5 (MEL-4.5) winch	ВАД-42	4.2	8.4	50	50	25

Machine or mechanism	Motor type designation	Nominal value			Overcurrent relay setting, A	Fuse link current rating, A
		Power, kW	Rated current, A	Starting current, A		
МЭЛ-11.4 (MEL-11.4) winch	MA-141-1/4	11.4	23	131	140	60
Prokhodka-500 fan . .	Special	9	19	130	130	60
Prokhodka-600 fan . .	Special	30	60	360	360	160
СБМ-3у (SBM-3u) boring machine . .	MA-143-2/4	16	32	190	200	80

The smallest size of cables for conveyors should range from 10 to 16 mm². Cables of smaller size have insufficient mechanical strength.

4. Determine the size of the armoured cable feeder run from the district substation to the local distribution centre. This is done by adding together the current ratings of all motors and devices which operate simultaneously at any given instant and selecting the armoured cable size from Table 17 for the aggregate current.

5. Calculate the total voltage drop in the circuit for normal running conditions of all the motors, with the drop across the transformer taken from Table 18.

Table 18 has been drawn up for full (100%) load. If the transformer carries only 50 per cent rated load, the value in the table must be reduced accordingly, and so on for other loads. For practical purposes, the power factor may be assumed to be 0.7 to 0.85.

The voltage drops in trailing and armoured cables are given in Tables 19 and 20.

Since the voltage drops in Tables 19 and 20 are given for a 100-metre length of cable, they must be multiplied by the respective factor to obtain the full drop in the actual length of cable. For example, an armoured 70-mm² cable, 300 metres long, feeds a coalcutter with an installed motor capacity of 47 kW and four conveyors driven by 11.4 kW motors. The coalcutter and three conveyors operate simultaneously. It is necessary to find the voltage drop in the armoured cable.

We first sum up the power taken by the motors running at the same time

$$47 + 3 \times 11.4 = 81.2 \text{ kW.}$$

Table 17

Fuse link current rating, A	Minimum permissible two-phase short-circuit current, A	Minimum permissible cable core size, mm²		Maximum continuous current-carrying capacity of cable, A	
		Rubber-covered cables ГРШ (GRSh) and ГРШН (GRShN)	Armoured cables СБ (SB) and СБГ (SBG)	Rubber-covered cables ГРШ (GRSh) and ГРШН (GRShN)	Armoured cables СБ (SB) and СБГ (SBG)
Rated voltages 380 and 660V					
20	140	2.5	—	25	—
25	175	2.5	—	25	—
35	245	4	2.5	34	28
60	420	6	4	43	37
80	560	10	6	55	45
100	700	16	10	70	60
125	800	25	16	95	80
160	800	35	25	115	105
200	800	50 (35)	35	145	125
—	—	—	50	—	155
—	—	—	70	—	200
—	—	—	95	—	245
Rated voltage 127 V					
6	24	2.5	—	25	—
10	40	2.5	—	25	—
15	60	2.5	—	25	—
20	80	2.5	—	25	—
35	140	4	—	34	—
60	240	6	—	43	—

Note: The cable core size shown in brackets, though permissible with respect to short-circuit current, is not desirable because of large voltage drop at starting.

Table 18

Transformer kVA rating	Voltage drop (volts) for power factors		
	0.7	0.8	0.85
75	21.0	19.8	18.8
100	20.7	19.6	18.6
180	20.4	19.2	18.2

For the nearest value, 80 kW and the size 70 mm², the table gives a voltage drop of 6.7V for 100 metres of cable. As the cable is 300 metres long, the total drop is

$$6.7 \times 3 = 20.1 \text{ V.}$$

6. Find the sum of the voltage drops in the transformer, trailing and armoured cables. The sum must not exceed the permissible value. If the total is too great, the size of the armoured cable must be increased, or the district substation moved up closer.

Table 19

Trailing cable size, mm²	Voltage drop, V, in supplying cutter-loader or coalcutter motor		
	МАД-191/11; МАД-191/11м		MA-191/10
	With 35 kW loader	With 13 kW loader	
25	—	—	10.5
35	—	12.3	7.5
50	11	8.6	5.5

Table 20

Load, kW	Voltage drop, V, for armoured cable sizes, mm²				
	25	35	50	70	95
20	4.7	3.3	2.3	1.7	1.2
30	7.0	5.0	3.5	2.5	1.9
40	9.4	6.7	4.7	3.3	2.5
50	11.7	8.4	5.8	4.2	3.1
60	14.0	10	7.0	5.0	3.7
70	16.4	11.7	8.2	5.8	4.3
80	—	13.4	9.4	6.7	5.0
90	—	—	10.5	7.5	5.5
100	—	—	11.7	8.4	6.2
110	—	—	—	9.2	6.8
120	—	—	—	10	7.4

7. Now determine the total voltage drop in the circuit when the coalcutter or the cutter-loader motor is being started and the other motors are running under normal load. For the drop in the transformer, Table 21 is used. It gives the drops according to the transformer rating, the type of the motor started and the value of the load current due to the other connected units.

Table 22 gives the voltage drop (per 100 metres) in the armoured cable, depending on core size, the type of the motor started, and the magnitude of the remaining load. The drops per 100 metres of trailing cables can be found from Table 23.

Table 21

Current of remaining load (conveyors, fans, pumps, etc.), A	Voltage drop, V				
	Type of motor				
	МАД-191/11 and МАД 191/11 м		МА-191 10		
	Transformer rating, kVA				
	100	180	75	100	180
25	55.4	31.4	69.4	53.4	30
50	58.6	33.2	74.4	56.8	32.2
75	62	35.2	78.6	60	34
100	65.5	37	—	63.6	36
125	69	39	—	67	38
150	—	40.8	—	—	40
175	—	42.7	—	—	41.8
200	—	44.6	—	—	43.7

Table 22

Current of remaining load (conveyor, fan, drill, etc.), A	Voltage drop, V			
	Type of motor			
	МАД-191/11, МАД-191/11м and МА-191/10			
	Armoured cable core size, mm^2			
	35	50	70	95
25	18.2	12.8	9.1	6.7
50	20	14	10.1	7.4
75	22	15.4	11	8.1
100	24	16.8	12	8.9
125	25.8	18.0	12.9	9.5
150	27.8	19.5	13.8	10.3
175	29.6	20.8	14.8	10.9
200	31.6	22.2	15.8	11.7

8. Add together the drops of the transformer, the armoured and trailing cables. The sum should not exceed the permissible value.

Table 23

Flexible cable size, mm²	Type of motor
	МАД-191/11, МАД-191/11м and MA-191/10
25	—
35	16.6
50	11.7

9. Finally, determine the current ratings of the fuse links and the current settings of the overcurrent relays.

Example 18. Given: the power distribution diagram of a mine district (Fig. 240).
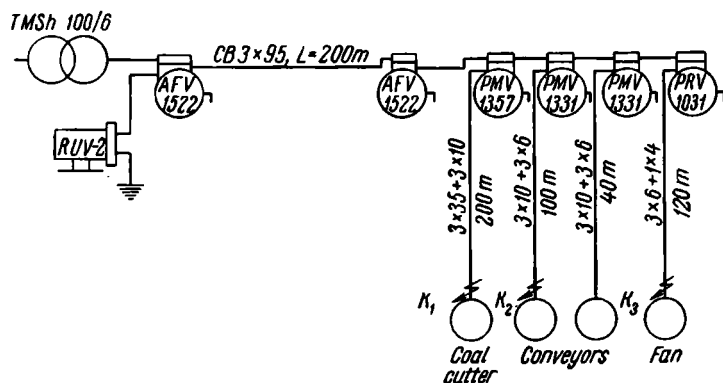
Fig. 240. District distribution circuit (for example 18)

1. Calculate the rating of the substation transformer.
2. Determine the sizes of the trailing and armoured cables according to their permissible current-carrying capacity.
3. Check the selected circuit for voltage drop under normal running conditions and when the coalcutter motor is being started.
4. Select the fuse links to be used in the gate-end starter boxes.
Data on the motors used in the district equipment may be taken from Table 16.
The coalcutter motor carries 75 per cent load; all the other motors are fully loaded.
1. The necessary rating of the district substation transformer is found on the basis of the total current load of the motors:

$$I = 96 \times 0.75 + 2 \times 23 + 19 = 137 \text{ A},$$

where
96A=current of the coalcutter motor for the 60-min rating;
0.75=motor load factor;
23A=conveyor motor current;
19A=fan motor current.

For the above load current, a 100kVA, ТМШ-100,6 (TMSh-100 6) transformer may be taken with a rated current of 145 A (since 145>137).

2. The sizes of the trailing cables are taken from Table 17 as follows: *for the coalcutter*, a six-core ГРШС 3✕35+3✕10 mm² cable having a current-carrying capacity of 115A (which is more than the motor rated current of 96A); *for the conveyors* (rated current 23A), a ГРШС 3✕10+3✕6 mm² cable. A cable cf 2.5 mm size with a current-carrying capacity of 25A (see Table 17) could cope with such a current, but its mechanical strength is too low.

A six-core cable has been selected to provide for remote control of the conveyors. For the fan, a ГРШС 3✕6+1✕4 mm² cable is selected.

3. Using Table 17 again, the armoured cable size is selected for the load current $I = 137A$.

A cable with 50-mm² cores capable of carrying 155A is taken to handle the 137A load current.

4. Next the circuit voltage drop is found for normal operating conditions. At a power factor of 0.8 the drop in the transformer (Table 18) is 19.6 V. Since the transformer is only loaded to $\frac{137}{145}$ 100=95% of its current rating, the actual drop in the transformer is $19.6 \times 0.95 = 18.6V$.

The voltage drop per 100 metres of trailing cable is 7.5V (Table 19). For the actual load of the motor and 200 metres length of the cable, the voltage drop will be

$$2 \times 0.75 \times 7.5 = 11.3 \text{ V}.$$

The drop per 100 metres of armoured cable (Table 20), is found for the aggregate power load:

$$P = 0.75 \times 47 + 2 \times 11.4 + 9 = 67.1 \text{ kW},$$

and a cable size of 50 mm².

The voltage drop (for the closest load of 70 kW) is 8.2 volts.

Taking into account the actual load and the armoured cable length, 200 metres, the true voltage drop will be

$$8.2 \left(\frac{67.1}{70}\right) \times 2 = 15.7 \text{ V}.$$

5. The total drop of the circuit is $18.6 + 11.3 + 15.7 = 45.6$ V which is in excess of the permissible value (39V).

Now the permissible voltage drop is found for the armoured cable 200 metres long:

$$39 - (18.6 + 11.3) = 9.1 \text{ V}.$$

Therefore, at a load of 67.1 kW, the loss per 100 metres becomes $\frac{9.1}{2} = 4.55$ V and corresponds to that of a 95-mm² cable (see Table 20), for which the drop is 4.3 V at a load of 70 kW. The cable to be used will then be СБ(SB) 3✕95 mm², although a 3✕50 size might have been sufficient to carry the load. This choice keeps the voltage drop in the circuit within the 39-V limit.

6. The voltage drop in the circuit when the coalcutter motor is being started with the remaining units running is now determined. The current taken by the latter is

$$2 \times 23 + 19 = 65 \text{ A}.$$

The drop in the transformer for the closest load of 75A is 60V (see Table 21). For 100 metres of 95 mm² armoured cable, the drop will be 8.1V (Table 22), and for 200 metres 2×8.1 or 16.2V.

For 100 metres of trailing cable (Table 23) the drop will be 16.6 V, while for 200 metres it will be 2×16.6 or 33.2V.

7. The resultant total voltage drop of the circuit at starting* will be:

$$60 + 16.2 + 33.2 = 109.4 \text{ V.}$$

The permissible voltage drop at starting of an MA-191/10 motor may be taken equal to $400 - 255 = 145$V.

Since the drop of 109.4V is within the permissible range, the circuit will maintain an adequate voltage at the motor terminals at starting.

8. The fuse links to be used in the gate-end starter boxes is now selected.

For a coalcutter with a motor rated for 47 kW, Table 16 gives 200-A links. This rating is checked for compatibility with the available size of trailing cable against Table 17. The highest rating of fuse links to protect a 35-mm² cable is 200A (see size 35 mm² in brackets and the note to Table 17). 200-A links should be used because they will be taken by the ПMB-1357 (PMV-1357) starter box.

A 200-A fuse link will reliably protect a cable if the two-phase short-circuit current will not be less than 800A (see Table 17). We therefore must find the two-phase short-circuit current in the trailing cable at point K_1 adjacent to the cutter.

The equivalent length of the 200-metre, 95 mm² armoured cable and the 200-metre, 35 mm² trailing cable (see Fig. 240) is

$$L_{eq} = 200 \times 0.54 + 200 \times 1.41 = 390 \text{ m;}$$

where the factors 0.54 and 1.41 have been taken from Fig. 181. For this equivalent length the alignment chart in Fig. 181 gives a two-phase short-circuit current of 850A, which is greater than the minimum 800A.

The fuse will definitely blow out on a short circuit in the trailing cable of the coalcutter.

60-A fuses should be used for the conveyors (Table 16); they will protect trailing cables of 6 mm² and greater size (Table 17), while in our case the cable is of 10 mm² size. Since a ПMB-1331 (PMV-1331) starter will take 60-A fuses, we should check whether they can blow on a short circuit at the far end of the trailing cable of the first conveyor (point K_2). The equivalent cable length to the fault is:

$$L_{eq} = 200 \times 0.54 + 100 \times 4.92 = 600 \text{ m.}$$

For this length the two-phase short-circuit current will be 640A, which is greater than the minimum value of 420A from Table 17.

For a fan driven by a 9-kW motor Table 16 gives a 60-A fuse, while the size of the cable should be at least 6 mm² (Table 16). The ПPB-1031 (PRV 1031) starter box permits use of 60A fuses.

The equivalent length to point K_3 is

$$L_{eq} = 200 \times 0.54 + 120 \times 8.22 = 1095 \text{ m.}$$

The two-phase short-circuit current for this case is 400A, which is less than the minimum value of 420A given by Table 17. Therefore, the size of the cable to the fan should be 10 mm².

Then $L_{eq} = 200 \times 0.54 + 120 \times 4.92 = 698$ m, for which the short-circuit current rises to 570 A and exceeds 420 A.

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* The total voltage drop at starting, as taken from the tables, will only be approximate because the actual load (65A) differs somewhat from the table load (75A).

Chapter X

FLAMEPROOF MOTORS

10-1. Series MA-140 Squirrel-cage Induction Motors

Series MA-140 flameproof squirrel-cage motors are used in underground installations to drive stationary equipment not subjected to heavy shocks or vibration. These motors are of the fan-cooled type in which the fan is seated on the shaft extension on the side opposite the drive end and blows the air over the surface of the motor frame (stator). The air inside the motor is circulated by blower vanes cast integral with the short-circuiting end rings of the rotor cage. Motors in this series are designed only for horizontal mounting and are available with one shaft extension. Typical applications for these motors are stationary conveyor and winch drives. They are not suitable for mobile machinery.

10-2. Series MA Slip-ring Induction Motors

The motors in this series are used to drive winches. Their slip rings are housed in an external flameproof enclosure. Connection to the starting resistors is by continuously-riding brushes. They are blast-cooled by a fan protected by a pressed-steel cowl. The cable boxes can take both trailing and armoured cables. This series is designed for operation on both 220V and 380V, except MA-36-71,4, MA-36-72/4 and MA-36-72/6 motors rated only for 380V.

10-3. The Kuzbass Series Motors

The Kuzbass flameproof series motors have been designed to drive stationary and transportable machinery in place of the MA-140 and MA-170 series motors. They are rated for 380 volts and have a star-connected stator winding.

Kuzbass motors are manufactured in two modifications:

(a) Fan-cooled (KO), with an external fan;

(b) Non-fan-cooled (K).

In both modifications the windings have heat-resistant varnished glass fibre insulation.

The stator frame (Fig. 241) is a smooth heavy-wall steel tube *1*, while the endshields *2* are machined at the outside edge to receive locking half-rings *3* which simultaneously fit into slots in the frame. Solid rings *4* serve to clamp the half-rings in place with studs *5*. Each ball bearing *6* is secured in the endshield by two housing caps *7* pulled

(a)

(b)

Fig. 241. General view (*a*) and longitudinal cross section (*b*) of a KO electric motor

tight by studs *8*. To prevent the grease from leaking out of the bearings, the bearing caps are fitted with felt seals *9*. The stator core *10* and the rotor core *11* are assembled of electrical-sheet steel stampings. Glass-fibre-base insulation is used for insulating the stator

winding. The rotor squirrel cage is of the cast aluminium type having integrally-cast blower vanes *13* on the short-circuiting end rings.

The air blast for external cooling is provided by a fan *14*, enclosed by a pressed-steel cowl *15* with air ports *16*. For mounting, these motors are provided with feet *17*. The terminal box is joined to the frame by a connecting sleeve and is fitted with a cable sealing end.

The K modification of motor is similar to the above KO motor, but has no external fan.

In addition to their main modifications, the Kuzbass series motors are available for various mounting conditions: КФ and КОФ for flange mounting, K and KO for foot-mounting, and a modification for both flange and foot-mounting.

Except for weight, the КФ and КОФ flanged motors have the same ratings as the respective K and KO motors.

The former ТАГ (TAG) series has been replaced by the KOM flameproof induction motors ranging from 0.6 to 7 kW in rating.

These are fan-cooled motors with an external fan fitted on the shaft and protected by a steel guide cowl. Their windings have Class B heat-resistant insulation with coils of ПЭТСО (PETSO) wire. The slot insulation is varnished glass-cloth and glass-micanite. The stator leads are brought out to a terminal box panel of special design in the form of a steel plate encapsulated in arc-proof plastic.

10-4. Coalcutter and Cutter-loader Motors

The type designations, ratings and other characteristics of the motors of coalcutters and cutter-loaders are given in Table 24.

The design features of the motors given in Table 24 are covered in the descriptions of the respective mining units.

The ЭКО (EKO) motors make up a new series of flameproof cutter-loader motors developed to provide better mechanical characteristics and highly reliable service. The motors in this series have open-type slots and rigid coils with silicone insulation. The rotor cage may be either of cast aluminium or welded-copper-bar construction.

The fan provided on these motors for external cooling has upgraded their continuous ratings two to three times compared with existing types.

10-5. Motor Operation and Maintenance

Before a motor is put into service, it must be examined to see that no foreign objects have been left inside it; the bearings must be cleaned and washed with kerosene and packed with grease; the air gap checked for uniformity, the rotor turned by hand to be sure it is free

Table 24

Motor type designation	Application	Modification	kW rating	Voltage rating, V	Speed, rpm	Stator rated current, A	Efficiency, %	Power factor	I_{start}/I_{rated}	T_{start}/T_{rated}	$T_{pul\text{-}tout}/T_{rated}$	Weight, kg
МАД-191/35г (MAD-191/35g)	For drive of ПМГ (PMG) coalcutters and loaders of all cutter-loaders	Non-fan-cooled	$\dfrac{15}{35}$	380	$\dfrac{1470}{1410}$	$\dfrac{40.5}{77}$	$\dfrac{88}{86}$	$\dfrac{0.63}{0.80}$	$\dfrac{8.5}{5.4}$	$\dfrac{6.5}{2.8}$	$\dfrac{6}{2.5}$	550
ЭДК-3-2 (EDK-3-2)	As above	As above	$\dfrac{14}{42}$	380	1500 (synch)	—	—	—	—	—	—	700
МА-191,10	For drive of КМП (KMP) coalcutters and МК and УКР (UKR) cutter-loaders	As above	$\dfrac{15}{47}$	380	$\dfrac{1485}{1455}$	$\dfrac{44.5}{96}$	$\dfrac{85}{89}$	$\dfrac{0.60}{0.83}$	$\dfrac{11.5}{5.8}$	$\dfrac{7.6}{2.3}$	$\dfrac{6.7}{2.1}$	840
ЭДК-3.5-1 (EDK-3.5-1)	As above	As above	$\dfrac{15}{58}$	380	—	—	—	—	—	—	—	950
ЭДК-3.5-2 (EDK-3.5-2)	As above	As above	$\dfrac{23}{65}$	380	—	—	—	—	—	—	—	1000
МАД-191/11м (MAD-191/11m)	For drive of cutter-loaders: Don-bass-1, ДУ-1 (DU-1), ЛГД (LGD), Gornyak, Kirovets	As above	$\dfrac{28.5}{65}$	380	$\dfrac{1485}{1455}$	$\dfrac{61}{136}$	$\dfrac{90}{89}$	$\dfrac{0.77}{0.82}$	$\dfrac{8.5}{3.8}$	$\dfrac{5}{2.2}$	$\dfrac{3.2}{1.4}$	1000
МАД-191,11мг (MAD-191/11mg)	As above	As above	$\dfrac{28.5}{65}$	380	$\dfrac{1485}{1455}$	$\dfrac{61}{136}$	$\dfrac{90}{89}$	$\dfrac{0.77}{0.82}$	$\dfrac{8.5}{3.8}$	$\dfrac{5}{2.5}$	$\dfrac{3.2}{1.4}$	1000

Motor type designation	Application	Modification	kW rating	Voltage rating V	Speed, rpm	Stator rated current, A	Effici-ency, %	Power factor	I_{start}/I_{rated}	T_{start}/T_{rated}	$T_{pull-out}/T_{rated}$	Weight, kg
ЭДК-4-1 (EDK-4-1)	For drive of cutter-loaders: Donbass-1, ДУ-1 (DU-1), ЛГД (LGD), Gornyak, Kirovets	Non-fan-cooled	$\frac{38}{88}$	380	$\frac{1490}{1470}$	$\frac{71}{171}$	$\frac{91}{92}$	$\frac{0.77}{0.82}$	$\frac{9.5}{4.2}$	$\frac{6.0}{2.6}$	$\frac{3.65}{1.58}$	1100
ЭДКО-4-1с (EDKO-4-1s)	As above	Fan-cooled	$\frac{62}{82}$	380	1500 (synch)	—	—	—	—	—	—	1100
ЭДКО-4-2с (EDKO-4-2s)	As above	As above	$\frac{77}{107}$	380	1500 (synch)	—	—	—	—	—	—	1165
ЭДК-120 (EDK-120)	For drive of cutter-loaders: Donbass-2, Donbass-6, Ukraina, УМК (UMK), УДК (UDK)	Non-fan-cooled	$\frac{50}{130}$	660	$\frac{1485}{1460}$	$\frac{61}{131}$	$\frac{91}{9}$	$\frac{0.77}{0.85}$	$\frac{9.7}{4.2}$	$\frac{5.5}{2.1}$	$\frac{4.9}{1.85}$	1800

Motor Troubles, Their Causes and Remedies

Trouble	Cause	Remedy
Stator winding hot throughout	1. Excessive load on motor or defective ventilation	Reduce load, blow out ventilating ducts
	2. Voltage too low at motor terminals	Restore rated voltage at motor terminals
Hot spots on stator winding. Motor hums, unbalanced currents in phases	1. Turn-to-turn short circuit in stator winding	Take motor out of service for repair
	2. Coils improperly connected in one phase	
	3. Short circuit between two phases	
At no-load, slip-ring motor starts up with its rotor circuit still open. When started under load, motor comes up to speed slowly, rotor runs hot	Short between neighbouring clips on end connections of rotor winding, or rotor winding is shorted at two places	Take motor out of service for repair
Rotor, and sometimes stator, run hot. Motor hums, comes up to speed slowly under load, fails to attain full speed	1. Bad contact in rotor circuit, or at connections between rotor winding and slip rings	Check connections of winding to rotor and resistor terminals
	2. Poor contact in starting resistor circuit	Clean contacts and rheostat brushes
	3. Broken rotor cage bars due to casting fault	Replace motor
Motor fails to start	No current in stator winding due to blown fuses or tripped circuit breaker	Replace fuses, reclose circuit breaker
Motor will not start; when turned by hand, runs with jerks and hums, no current in one phase	Break in one phase of supply circuit or in stator winding	Check voltage at all motor terminals. If break is in supply phase, or line voltages differ (for example, a fuse has blown out), eliminate fault
Motor fails to start with rated voltage at its terminals	1. Break in two (or three) phases of starting rheostat or in wiring between rotor and rheostat	Locate and eliminate fault
	2. One-sided pull on rotor due to worn bearing or shifted endshield	Check air gap between stator and rotor
Rotor rubs against stator	1. Worn bearings, shifted endshields, bent shaft, deformed rotor or stator core	Check air gap between stator and rotor. Examine shaft. Take motor out for repair if cores are deformed
	2. Unbalanced rotor	Take motor out for repair

to rotate, and the shaft inspected for sufficient end play. The fixing, bearing and terminal bolts must be checked for tightness, the motor leads inspected for proper connection to the circuit, the frame checked for reliable earthing, and the protective and signalling devices also examined for positive action.

The motor should then be run at no-load to determine how its mechanical and electrical parts operate, whether the bearings run without heating, and if the direction of rotation is correct.

When a motor runs under load, the temperature rise of its winding and core must be watched. The temperature must not exceed the limits prescribed for the class of insulation used in the motor. The maximum safe temperature for antifriction bearings is 130° C. During service, the motor should be regularly checked for the size of its air gap, loose bolted joints, excessive vibration, and sufficient grease in the bearings. The insulation resistance of the winding should also be measured and not be under 1000 ohms per volt of motor rating (for a 380-volt motor the insulation resistance should exceed 380,000 ohms).

The operation and care of motors is the duty of the operators of the units on which they are installed and the district electrical fitters.

Any fault occurring during a shift must immediately be reported by the electrical fitter to the shift mechanic or the district chief.

The motors of face machinery should be regularly inspected at intervals of not more than one month by a maintenance team headed by the district mechanic.

During maintenance inspections, it must be seen that the bearings are good and have enough grease in them, that the rotor does not rub against the stator, the motor is properly coupled to the gear box of the machine, and that none of the mounting bolts and terminal nuts have become loose. The insulation resistance of the windings relative to the motor frame should be measured also.

Routine maintenance repairs are made on the spot. This includes full inspection and a more thorough examination of the bearings and motor parts.

Major repairs and overhauls must be made in special repair shops.

C h a p t e r XI

UNDERGROUND LIGHTING

11-1. General

The introduction of new and complicated machinery in coal mines has imposed more stringent requirements on underground electric lighting. The former types of stationary lighting fittings and portable battery lamps have proved inadequate for proper operation and care of the new machinery.

In recent years Soviet scientists and designers have developed new portable battery lamps, and stationary lighting fittings using incandescent and fluorescent lamps, now manufactured commercially.

Adequate lighting is a major factor ensuring the safety of men working underground. Furthermore, it is conductive to higher work standards, higher productivity, improved working conditions, and stimulates the men to maintain their work places cleared up.

Mine electric lighting systems must be installed so that they meet the requirements imposed by the mine atmosphere, the presence of gas, moisture and dust, and the dangers of mechanical injury by falling lumps of rock and coal.

11-2. Portable Battery Lamps

The main sources of light at the face are the individual portable storage-battery safety lamps.

They may be of the cap and the hand type.

A cap lamp has a headpiece for attachment to the helmet of the miner, with the storage battery attached to his belt, leaving both hands free.

A hand safety lamp comprises a casing which houses a storage battery and a cap fitting which is screwed into the casing and contains

the lamp bulb. The cap has four support pins to carry a cover plate and the carrying hook. Two- and three-cell alkaline storage batteries are generally used as the sources of supply in portable lamps.

In these storage batteries the plates may be of two types. In the first type the active material of the plates are enclosed in perforated steel pocket envelopes; in the second type, the powdered active materials are pressed into shape and covered with a thin insulating cloth pervious to the electrolyte.

Batteries without pocket envelopes have a considerably higher (40 to 50%) amperehour capacity per unit of weight and volume, and contain a limited amount of electrolyte, thereby eliminating any spillage. They can be recharged without unscrewing the plugs and be topped-up with electrolyte after every 5 to 7 charge-discharge cycles. However, their service life is 2 to 2.5 times shorter than that of batteries with pocket envelopes.

The batteries are charged with direct current at 1.8 to 1.9 volts per cell. The electrolyte is a solution of sodium or potassium hydroxide to which a little lithium hydroxide is added; the lack of this additive will shorten the service life of the battery (from 2.5 to 3 times).

The cap lamps in use at the present time in the U.S.S.R. are: the Ukraina, ШГС-1 (ShGS-1), ШГС-3 (ShGS-3), and ЛСК-10у (LSK-10u); the hand lamps in use are the ЛАУ-4 (LAU-4) and ЛАТ-2 (LAT-2).

A battery cap lamp (Fig. 242) consists of a sheet steel case *1* for the battery, a case cover *2* fitted with spring contacts to make connection with the battery, a headpiece *3* containing a lamp bulb, and a switch *4*. The headpiece is connected to the battery case cover through a two-conductor flexible cord *5* and the case is fitted with loops so that a waist belt can be passed through them. The light bulb contains a working and a pilot filament which can be switched on at will. The main filament furnishes light when working, the pilot filament when walking in workings.

The lamp is mounted in a vertical position within the headpiece and is held in the holder contacts by a clamp spring pressing against the bulb. Therefore, should the protective lens or bulb be broken, the headpiece is de-energized. A strong transparent plastic lens is used to protect the bulb.

Ukraina cap lamps employ an alkaline nickel-zinc battery of semi-dry construction carried in a rubber-doped polystyrene case. A metal cover is hinged to the battery case. The headpiece is of fibre-reinforced plastic, contains a chromium-plated reflector and carries the lamp bulb in the centre. Topping-up of the battery is done every 9 to 10 days with a solution of sodium hydroxide in distilled water. In the older ЛСК-10у (LSK-10u) cap lamps topping-up is necessary every day.

The battery contains two cells and is charged through the headpiece without any disassembly of the lamp on a charging rack, thus making it suitable for self-service system lamp rooms. Under this system, daily dismantling and reassembly is dispensed with and some of the maintenance operations are performed by the workmen.
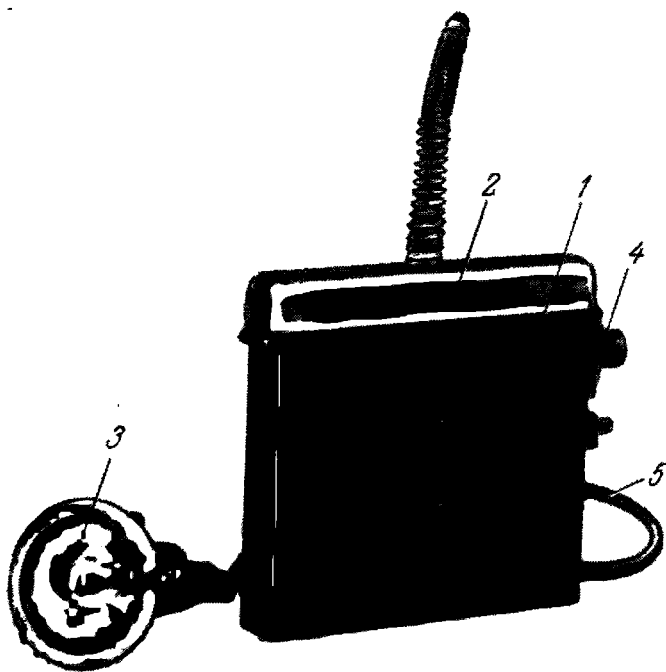
Fig. 242. The ЛСК-10у (LSK-10u) battery cap lamp

The ШГС-1 (ShGS-1) cap lamps have a three-cell nickel-iron battery without pocket envelopes. In them the cells are insulated from each other by rubber coverings. The battery does not require daily topping-up and may even operate when the electrolyte is spilled out, due to the fact that its main portion is absorbed by the active material of the plates. Charging can be done without unscrewing the vent plugs, for which reason this head lamp is suitable for self-service maintenance.

A new headpiece with a small-size, two-filament, xenon-filled bulb has been developed for this lamp, eliminating the intricate and unreliable disconnecting device of the earlier models. The use of a strong transparent material makes the headpiece flameproof.
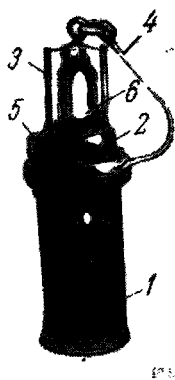
Fig. 243. ЛАУ-4(LAU-
4) battery hand lamp

The ШГС-3 (ShGS-3) cap lamp differs from the lamp above in that it has a plastic case, although it employs the same model of headpiece. The monolithic plastic case of this cap lamp carries a three-cell nickel-iron battery without pocket envelopes.

The ЛАУ-4 (LAU-4) battery hand lamp, designed for 360-degree light distribution (Fig. 243), comprises a case *1* accommodating a storage battery, a lamp head *2* and head post pins *3*, to which the cover plate and carrying hook *4* are fixed. Owing to a magnetic lock *5*, the head can be removed in the lamp room only. The bulb is protected by a glass cap *6*.

Hand lamps ЛАУ-4 (LAU-4) and ЛАТ-2 * (LAT-2) are not flameproof because the glass cap in the first lamp and a special de-energizing device in the second will not prevent ignition of fire damp, should the glass cap or the bulb break. Nevertheless, these lamps are permitted for use in gassy and dusty mines.

11-3. Lighting Fittings for Fixed Systems

Fixed or mains lighting systems for rooms, pit bottom and haulage roadways use the following types of incandescent lamp fittings:

(1) PH-60-1 (RN-60-1), PH-60-2 (RN-60-2), PH-100 (RN-100) and PH-200 (RN-200), of standard mine design, for installation in mines free from gas and dust hazards, and for installation in Class I and II gas and dust hazardous mines only in the intake air stream of main haulage roadways;

(2) PП-60 (RP-60), of higher-reliability design, is for installation in Class I and II gas and dust hazardous mines in all workings with the exception of blind headings ventilated by temporary fans.

The PH-60-1 (RN-60-1) fitting shown in Fig. 244 consists of a body *1*, protective wellglass *2* and wire guard *3* fixed to a pressed ring *4*. The latter makes a bayonet joint with the body and a rubber gasket is used to seal the joint between the body and the wellglass. A thrust screw *5* at the bottom of the fitting serves to press the wellglass against the body, the cable gland *6* takes a flexible cable, and a screw on the body provides for earthing.

The PH-60-2 (RN-60-2) fitting (Fig. 245) differs from the above in that it has two glands for entrance and exit of flexible cables to

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* Since 1959 superseded by the Model ЛАТ-4 (LAT-4) hand lamp.

neighbouring fittings. Where the supply comes from the locomotive trolley wire circuit, this fitting will take a 75W, 250V lamp.

PH-100 (RN-100) and PH-200 (RN-200) lighting fittings are used for lighting the pit bottom roadways, underground workshops, locomotive garages and servicing stations, machine rooms, and also the coal stores, spoil heaps and surface structures.

The PH-100 (RN-100) fitting (Fig. 246) comprises a body *1*, wellglass *2*, and wire guard *3*.

By means of the retaining ring, the wellglass is clamped to the body with a rubber gasket placed between it and the body to prevent ingress of coal dust. The fitting is earthed with a screw provided on the body. A single gland *4* is fitted for cable connection. This fitting takes a 100W lamp, while the PH-200 (RN-200) fitting is for lamps up to 200W.

The PⅡ-60 (RP-60) higher-reliability type lighting fittings (Fig. 247) are made with a special lampholder consisting of a body *1*, centre contact *2*, side contact *3*, lead-in terminals *4* and *5*, and an end ring *6*.

The lampholder is free to move on guide studs *7*. Springs *8* hold it tight against the lamp base, thus causing the lampholder contacts to thrust against the base contacts, while the lamp bulb, at the other end, bears on the protective wellglass.

Should the wellglass or the lamp bulb break, spring *9* will push the lamp socket out of the lampholder and open the circuit between contact *2* and terminal *5* in the flameproof space of the lampholder.

Three bosses on the lower internal spherical surface of the wellglass form the seat upon which the bulb rests.
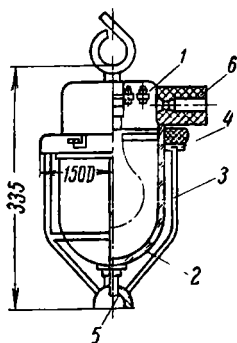
Fig. 244. PH-60-1 (RN-60-1)
mine lighting fitting
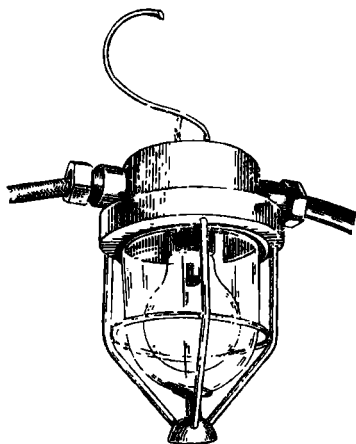
Fig. 245. PH-60-2 (RN-60-2) mine
lighting fitting
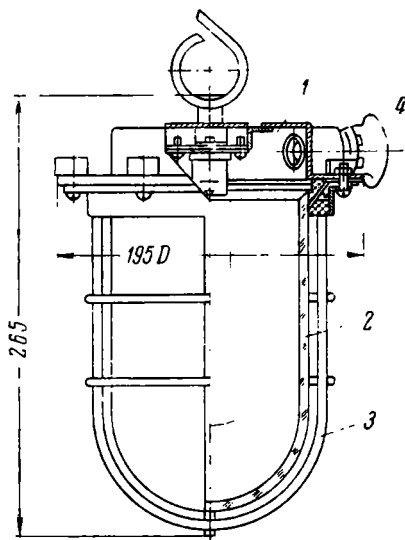
Fig. 246. PH-100 (RN-100) mine
lighting fitting
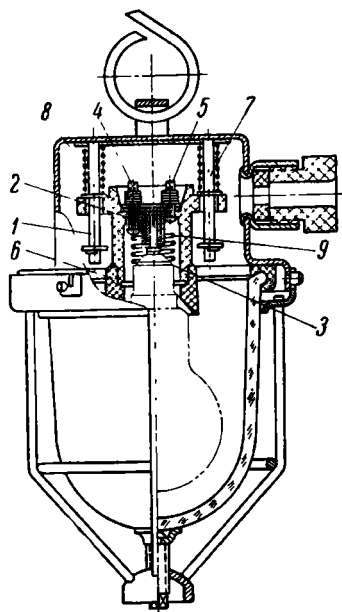
Fig. 247. РП-60 (RP-60) mine
lighting fitting

The wellglass has three grooves on the inside between the seating bosses. Cracks due to a blow will first develop at the grooves, and so there is a greater probability that the fitting will be de-energized.

11-4. Mine Fluorescent Fittings

The use of fluorescent lighting fittings in mine haulage roadways is widely practised in the U.S.S.R.

The fluorescent lamp consists of a long glass tube with tungsten electrodes sealed into each end. The tube is evaluated to a low pressure and contains a small amount of mercury vapour and argon gas. The inside surface of the tube is coated with a fluorescent compound, termed the phosphor. When a voltage is applied to the tube electrodes, they grow hot and a mercury-vapour arc strikes in the tube, emitting a large amount of invisible ultraviolet radiation which strikes the phosphor, causing it to fluoresce and emit visible light.

Since fluorescent lamp electrodes operate at a temperature much lower than that of an incandescent lamp, they cool so quickly that a firedamp atmosphere will not ignite; the lamp is therefore flameproof.

Fluorescent lamps of the same lumen output take about one-third of the power required by incandescent lamps and have a longer service life.

A voltage from 600 to 700V must be applied to the electrodes to start a fluorescent lamp, it therefore requires a special device for use on 127- and 220-V lighting circuits.

Fluorescent lamps may be started with either hot-cathode or cold-cathode action. Hot-cathode starting is called preheat starting, cold-cathode starting is called instant starting. Suitable starting circuits are used in either case.
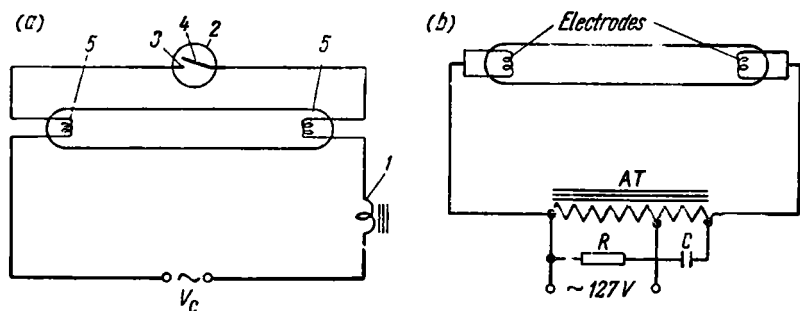
Fig. 248. Starting circuits for fluorescent lamps:
a—preheat starting; *b*—instant starting

The preheat-start lamps (Fig. 248*a*) are connected in series with a choke *1* and a gas discharge relay *2* (starter). When the lamp unit is switched on, the voltage across the starter electrodes *3* and *4* initiates a glow discharge in the starter which heats electrode *4*, causes it to bend until it comes in contact with electrode *3*, and thereby establishes the path for flow of current from the supply through lamp electrodes *5*. The electrodes reach a temperature of 1100 to 1200° C (normal operating temperature of electrodes being 800-850°C).

The glow discharge in the starter ceases as soon as its contacts meet; electrode *4* then cools down and acquires its former shape, breaking the circuit between the lamp electrodes and causing the choke to produce a high voltage surge between the hot lamp electrodes which fires the lamp.

A limitation of this circuit is that if the fluorescent tube is broken the arc between its electrodes instantly dies out, the starter again becomes operative, recloses the preheat circuit, and the hot electrodes may ignite the inflammable mine atmosphere.

The instant-start circuit (Fig. 248*b*) uses an autotransformer. When the 127-volt supply is applied to the autotransformer, the voltage across its end leads and, hence, across the lamp electrodes, rises to

14*

800-880 volts. This voltage strikes an arc through the argon gas and starts conduction, although the lamp electrodes are still cold.

To prevent the resultant flow of current from reaching an excessive value, the autotransformer is designed for high leakage reactance to lower the voltage applied to the lamp to the normal operating voltage. The capacitor incorporated in the circuit serves to improve the power factor.

Mines that are not gas or dust hazardous and also the main haulage roadways (in the intake air stream) of gas and dust hazardous mines use standard, general-purpose mine fluorescent fittings (Fig. 249).
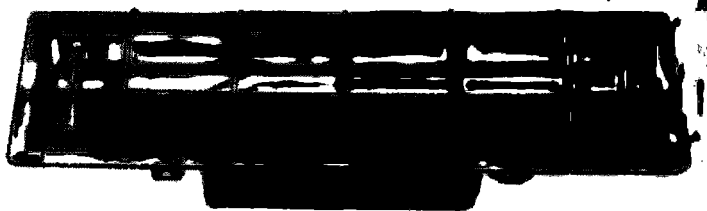
Fig. 249. РНЛ-15 (RNL-15) standard, general-purpose mine fluorescent lighting fitting

Such a fitting has a metal support body, two lampholders and a wire guard. The choke and the starter are mounted in a box fitted on top of the body.

Where safety requirements so dictate, flameproof РВЛ-15 (RVL-15) fluorescent fittings must be installed (Fig. 250).

The РВЛ-15 (RVL-15) fitting is a head carrying two cable glands, terminals for connecting the supply cable, and means for attachment of the fitting frame which takes a 15W fluorescent tube and includes the contact system and starting devices. By providing two cable glands, the need for a three-way cable box is eliminated. The connection between the head and fitting frame is of the plug-in type, with the frame jointed to the head by a coupling nut. To prevent the fitting from being opened by an unauthorized person, the nut is locked with a socket-head screw which can only be backed out with a special key.

The contact system is so designed that when the fitting frame is detached all the open conducting parts are de-energized.

Two-part construction (head and fitting frame) makes it a very simple matter to replace the faulty fluorescent tube or starting device in these fittings without disturbing the cables.

View A
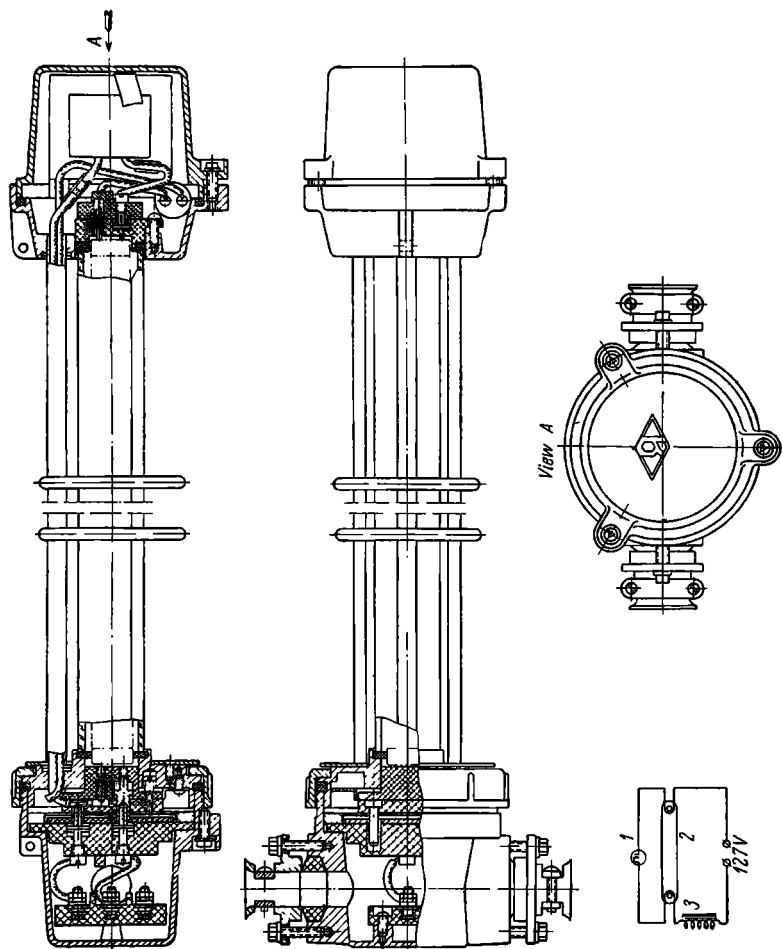
Fig. 250. РВЛ-15 (RVL-15) flameproof fluorescent lighting fitting:
1—starter; *2*—fluorescent tube; *3*—choke

127V

Mechanical interlocks instantaneously de-energize the fitting if the tube is fractured. The protective tube of this fitting is made of a transparent plastic having high mechanical strength, and the fitting as a whole is of dust-and-water tight design.

The conventional preheat-start circuit is used to start the lamps.

РВЛ-15 (RVL-15) fittings have replaced the outdated РВЛА-15 (RVLA-15) fittings of the instant-start type.

11-5. Components of Underground Lighting Installations

Stationary or mains lighting circuits are fed by 2.5- and 4-kVA ТСШ (TSSh) portable three-phase transformers with a 220,380V primary and a 1·33V secondary. Jumpers on the terminal panels permit the primary to be star-connected for 380V supply or delta-connected for 220V supply. A screw on the casing provides for connection of the cable earth core.

These are dry-type transformers with flameproof enclosures mounted on skids for portability. Another transformer finding application is the single-phase OC-1.5/0.5 (OS-1.5/0.5) dry-type 1.5kVA transformer, with a 380 or 220V primary and a 133V secondary.

Lighting transformers are connected to the supply through ПБГ-380 (PBG-380) manually operated starters.

Tee-off lines to lighting fittings are taken from the lighting-mains armoured cable by means of TM-10 flameproof three-way boxes. The latter will take three-core armoured cable mains in sizes up to 10 mm² and tee-off flexible cables up to 25 mm in outside diameter (size $3 \times 6 + 1 \times 4$ mm²). The box contacts are rated for up to 50A and 250V. Where the lighting mains use flexible cable, a flameproof TM-6 three-way box must be used for a tee-off. These boxes are designed to receive cables with outside diameters as great as 25 mm and will carry load currents up to 45A at voltages up to 250V.

11-6. Underground Lighting Circuit Arrangements

Trolley-wire locomotive haulage roadways may be lighted with two-gland PH-60-2 (RN-60-2) fittings carrying a special 250V lamp (250V the average voltage of the trolley wire). The fittings should be connected between the trolley wire and the rails, and spaced 10 to 20 m apart.

Where trolley-wire supply is not available in haulage roadways, and for lighting gravity haulages, rise headings and intake airways, a sectionalized system is practiced. Each main is built up of sections of flexible cable from 50 to 60 metres long joined together by CM-5 (SM-5) or CM-6 (SM-6) couplers, while the tee-offs to the lighting fittings are made with three-way boxes as shown in Fig. 251.

Power supply for the ТСШ (TSSh) lighting transformer is taken at 380V through a ПБГ-380 (PBG-380) manually-operated starter, the outgoing cable from the transformer secondary being connected to the main by a coupler [CM-5 (SM-5) or CM-6 (SM-6)]. Each mains section feeds three fittings connected as stated above.
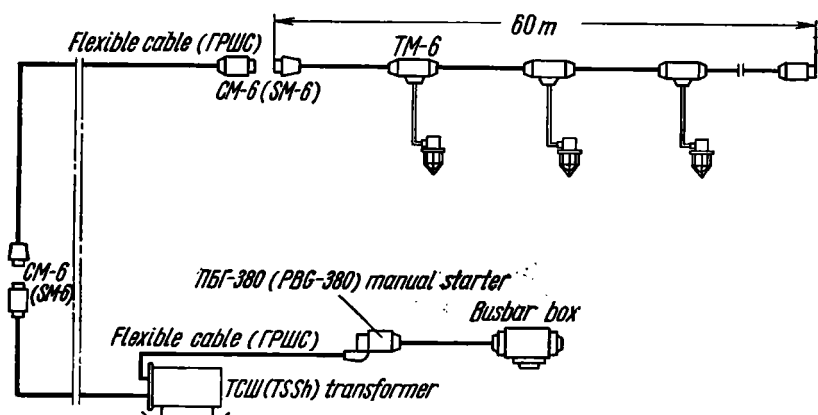
Fig. 251. Section-main layout of lighting fittings

Sectionalized mains are also employed at the faces of entry headings, each section having three fittings 6 metres apart and joined with plug and socket couplers. As the face is advanced, the main has to be lengthened because the lighting transformer remains at the gate-end distribution point. This is done by coupling on new sections of flexible cable.

11-7. Care of Underground Lighting Installations

The shift electrical fitter must keep constant watch on the lighting fittings, transformers, mains cables and starting devices.

Before any fitting is mounted, it must be examined for a close fit at the joint of the protective envelope and to see that all the necessary packings, bolts, etc., are in place. Earthing of the fitting at its body requires particular attention. The protective envelope should be kept clean of coal dust, with the bolts and clamping screws maintained tight and regularly checked. As a precaution against a cable being pulled out of the gland, the holder clips must be maintained tight.

Until the power has been cut off in a given section, no attempt should be made to examine or repair its cables, lighting fittings or cable accessories.

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11-8. Petrol Safety Lamps

Petrol safety lamps provide indication as to the presence of fire-damp in the mine atmosphere.

The ЛБШ (LBSh) safety lamp (Fig. 252) has a fuel reservoir and a cover carrying the wick holder and provided with a hole closed by a plug. Its upper part comprises a threaded brass ring supporting a set

Fig. 252. ЛБШ (LBSh) petrol safety lamp

of metal post rods to form a framework within which the inner and outer gauze cylinders are placed and rest on the glass shield. This lamp is fitted with magnetic and sealing locks. When the lamp is used in workings where the air stream velocity exceeds 5 metres per sec, the lamp is fitted with a steel guard hood.

The height of the blue halo or "gas cap" above the flame in these lamps is a measure of the per cent content of methane in the mine atmosphere.

Chapter XII

UNDERGROUND PROTECTIVE EARTHING

To operate properly and safely, electric machines and apparatus must have their current-conducting parts fully insulated from all parts having any connection with earth (frame, core). Any break-down of the insulation due to mechanical injury or electrical puncture results in a flow of current to earth (earth leakage current) through the fault and appearance of a dangerous potential between the frame and earth. Anyone coming in contact with such a frame will receive an electric shock dangerous to life. When a miner comes in contact with a gate-end box enclosure which has acquired potential due to a fault in the insulation of a current-conducting part (Fig. 253), the earth leakage current will have two paths for flow:

(a) through the enclosure, its skids and the box earthing lead;

(b) through the enclosure and the person in contact with it.

The less the resistance of the earthing, the larger the current flow-ing through it, and the less the current able to flow through the body of the person in contact with the box. A current of only 0.1A is fatal to human beings.

For safety from electric shock, the path for earth leakage currents must be made sufficiently conductive through the earthing and as difficult as possible through the human body. To accomplish this, it is necessary to earth (connect to earth) the enclosures of electrical equipment, the lead sheaths and armour of cables, the metal cable boxes and also all metal equipment having no relation to electrical equipment but liable to acquire a potential in the event of an electri-cal fault (piping systems, etc.). The path for electric current through the body is made difficult by using and wearing safety aids when operating live equipment (rubber mats, boots and rubber gloves).
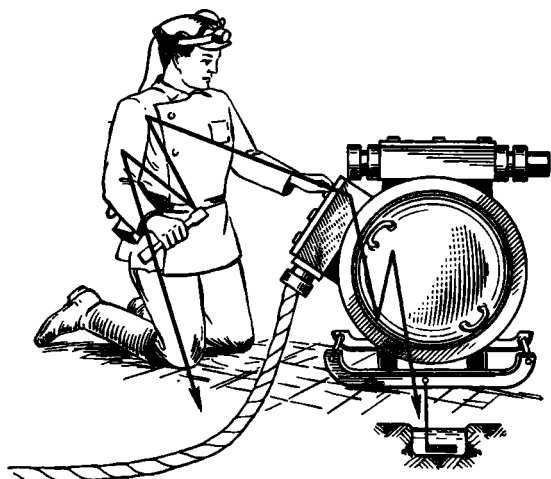
Fig. 253. Path of fault current to earth

prevail, the resistance of the human body is generally low (not over 1000 ohms), and the continuity and resistance of the earthing system therefore require constant and close control. This also applies to the safety aids.

12-1. The Earthing System

Each earthing circuit consists essentially of an earthing electrode system or assembly and the earthing conductors. The earthing electrode system comprises the conductors and electrodes brought in intimate contact with the soil to obtain a reliable electrical connection to earth. The resistance to the flow of a current to earth at the earthing system electrodes and conductors is called the earth resistance. The less its value, the better its quality. To lower it, the electrodes and conductors should be installed in damp locations, for example, along drainage ditches, sumps, etc.

Low overall earth resistance is attained by providing a general underground earthing circuit, for which the steel armour and lead sheaths of all the armoured cables are electrically connected to each

other and also to main earthing electrodes and conductors installed in a sump or water-drainage basin.

A mine must have at least two main earthing systems, each serving as a standby for the other during repair or cleaning. Another measure making for low overall earth resistance is the provision of local electrode systems at each unit or group of units of apparatus. This creates an integral and uninterrupted earthing circuit to which all the equipment requiring earthing, as well as the local and main electrode systems, are connected in parallel.

Pieces of equipment must never be earthed in series, because a break in the earthing lead will leave the equipment unearthed. A main earthing electrode is a steel plate at least 5 mm thick and 0.75 m² in area. A stationary electrical installation is earthed by placing a steel strip not less than 0.6 m² in area and 3 mm thick in a deepened portion of the gate drain ditch. Dry soil situations have to be coped with by using earthing electrodes made from pipes at least 1.5 m long and not less than 35 mm in diameter and placed in bored holes. The latter must have a greater diameter than the pipe and the space between the pipe and hole wall should be filled with sand or cinders. From time to time a solution of salt in water should be poured into the internal space of the pipes. The pipes should also have twenty or more 5 mm holes drilled in them at different heights. By this method the earth resistance is reduced to a satisfactory value.

For making the earthing conductor lead-out from the earthing electrode, steel strip not under 50 mm² in cross-sectional area is used, the jointing being done by welding where permissible and by bolting in gas and dust hazardous mines.

The U.S.S.R. safety regulations require that the earth resistance of the entire earthing circuit, as measured at the furthermost point, must not exceed 2 ohms.

When cables are dropped underground through bore holes from the surface, the main electrode earthing systems are put in at the surface or in one of the water-drainage shafts of the mine.

An additional earthing electrode is provided by the bore-hole pipe casing.

Hauled, portable and self-propelled mining equipment such as cutter-loaders, coalcutters, and hand and post-mounted electric drills are earthed by the earthing core incorporated in their trailing cables. One end of the core is connected to the earthing terminal of the starter box controlling the given machine or device, the other end to the frame of the machine or device.

Operators of transportable and self-propelled machinery are in constant contact with their frames and enclosures. This is why close attention must be paid to their earthing, because the danger of electric shock is especially high when the earthing is defective.
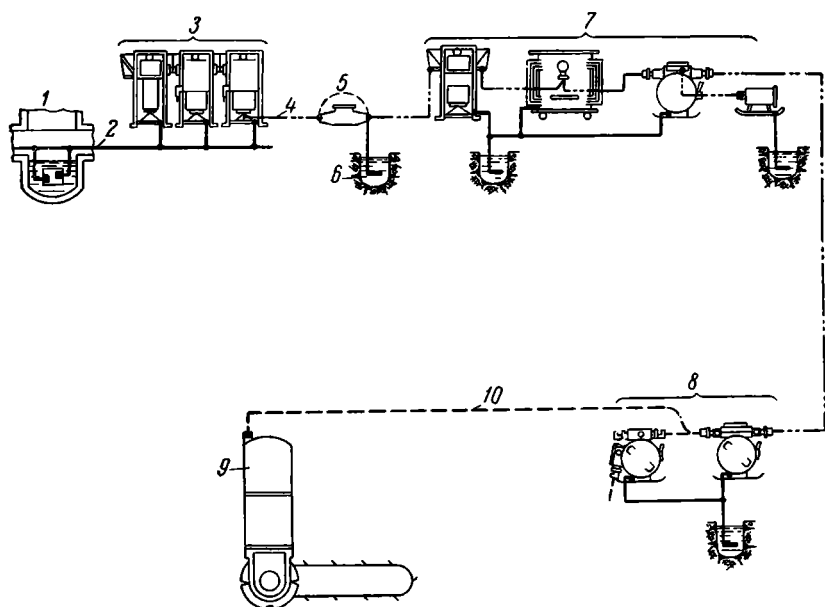
Fig. 254. Essential elements of an underground earthing circuit

The essential elements of an underground earthing circuit are represented in Fig. 254.

It shows that the main earthing electrode *1* is joined to earthing conductors *2* brought into underground substation *3*. All the enclosures of the apparatus (high-voltage switchgear, transformers, etc.) are connected to earthing conductors *2* in parallel. The enclosures of the apparatus also have the armour and lead sheaths of the high-voltage cables earthed to them. In the through-joint cable box *5* the iron casing is connected to the armour and lead sheath of cable *4* and is additionally provided with a local earthing *6*. For greater reliability, the incoming and outgoing cable ends within the box have their armour and lead sheaths interconnected by copper conductor jumpers. At the district substation *7* the high-voltage switchgear unit, transformer and the enclosures of the feeder circuit breakers are earthed by connecting them to the general underground earthing circuit through the armour and lead sheaths of the cables and to the local electrode system. The gate-end starting boxes at the district distribution centre *8* are earthed in a similar way.

The coalcutter *9* is earthed by its connection to the earth core *10* of the trailing cable. Thus, the main earthing electrode system and the local electrode systems are all connected in parallel and tied into an integral underground earthing circuit.

To earth an armoured cable, the lead sheath is cut, when making a connection for some length on two opposite sides of the cable end and turned backward over the steel armour. The contact surfaces of the armour and lead sheath must be cleaned bright.

After a cable box has been filled with compound, a steel clamp is attached over the brought-out lead sheath at the box for joining it with the earthing lead from the local earth electrode. This same lead is connected to the cable box by its clamping bolts. The clamp mentioned above should have a width of at least 25 mm.

The connection between the steel clamp and the cable box must be made with steel strip of not less than 50 mm² cross-sectional area or with a copper jumper conductor of at least 25 mm² size.

Motors are earthed by connecting the lead from the earthing circuit to the bolt specially provided on the frame for this purpose. Transformers are earthed with an earth lead clamped to the armour of the high-voltage supply cable at one end, and to the earthing bolt on the transformer enclosure at the other. This bolt also receives the second earth lead connected to the armour of the outgoing cable in the same way as on the supply cable.

A stationary distribution switchgear point is earthed by installing an earthing steel strip of at least 50 mm² cross-sectional area and connecting to it the leads from the earthing electrode and armour and sealing boxes of all the cables.

Gate-end distribution points, being of the portable type, are earthed by connecting the metal cable armour and sheaths to the gate-end boxes and each gate-end box to the local earthing system.

12-2. Measurement of Earth Resistance

Safety regulations require that the total earth resistance of the underground earthing circuit should not exceed 2 ohms as measured at the furthermost point. As the earth resistance depends on many factors, especially the moisture content of the soil, it must be measured regularly and at intervals not greater than one month. The higher the earth resistance the poorer its protective action when a breakdown occurs in the insulation of a machine or apparatus.

The instrument most extensively used in the U.S.S.R. for these measurements is the MC-07 (MS-07) earth tester (Fig. 255). This is a portable instrument with three ranges: 0-1000, 0-100 and 0-10 ohms. It can measure the earth resistance of earthing systems, the resistivity of soils and also measure the resistance of conductors from 0.02 up to 1000 ohms.

The tester has a voltage coil connected across the supply terminals and a current coil connected in series with the resistance to be measured. Therefore, this instrument measures resistance by the fall-of-

potential (voltmeter-ammeter) method. With the coils so connected, the pointer deflection is proportional to the resistance being measured. The test voltage is provided by the hand-operated d.c. generator built into the instrument.

The instrument measures $200 \times 245 \times 445$ mm overall and weighs 13.5 kg. To take a measurement, terminals E_1 and I_1 (Fig. 256) are
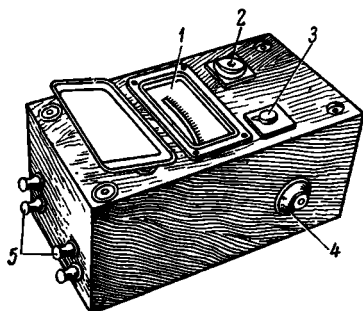
Fig. 255. MC-07 (MS-07) earth tester:

1—scale and pointer; 2—range selector switch; 3—"adjust-measure" changeover switch; 4—rheostat knob; 5—terminals for connection of earthing circuit to be tested and auxiliary earthing rod
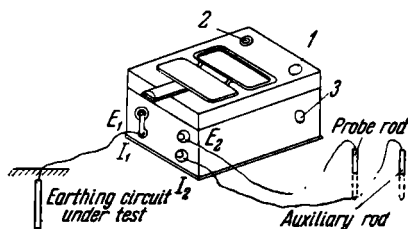
Fig. 256. Connections of MC-07 (MS-07) earth tester

made common and connected to the earth connection under test. Terminal E_2 is connected to the probe rod and terminal I_2 to an auxiliary earth rod. The probe and auxiliary rods may be steel bars or pipes driven into moistened ground to a depth more than 0.5 m.

If the resistance to earth of some local earthing electrode must be measured, it is disconnected from the general underground earthing circuit and connected to shorted terminals E_1 and I_1, following which terminal I_2 is connected to the general earthing circuit serving in this case as the auxiliary earth electrode. Terminal E_2 is connected to the probe electrode.

When the resistance to earth of the general earthing circuit must be measured, any one of the local earthing electrodes can be disconnected from it to serve as the auxiliary earth connection.

For taking a reading the "Adjust-measure" selector switch 1 is set in the "Adjust" position and rheostat knob 3 is turned while cranking the generator operating handle until the pointer reaches the red mark on the scale.

The switch is then thrown over to its "Measure" position and a reading is taken with the range switch 2 put in the "$\times 1$" position and the handle cranked at a speed of about 2 rps. If the resistance to

earth of the earthing under test proves to be less than 100 ohms, a more exact reading is obtained by turning the range switch *2* to the "×10" position. The reading obtained in this case should then be divided by ten.

Range switch *2* may be at once set in the "×100" position when measuring the resistance to earth of an underground earthing; in
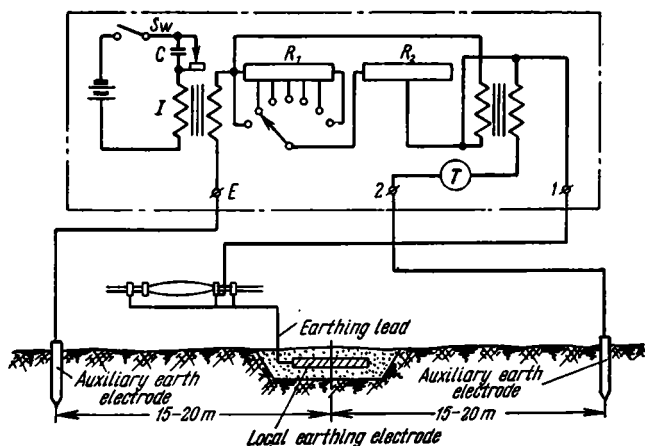
Fig. 257. Connections for measuring earth resistance with ИЗШ-52 (IZSh-52) earth tester

this case the resistance to earth will be equal to the scale reading divided by 100.

In addition to the MC-07 (MS-07) instrument, earth resistances may be measured with the ИЗШ-52 (IZSh-52) intrinsically-safe mine-type earth tester (Fig. 257).

The interrupter (buzzer) *I* in the tester converts direct into pulsating current which is sent through the circuit under test.

Rheostats R_1 and R_2, respectively of 1000 and 100 ohms, are used for adjustment during testing. Telephone *T* serves as the means for signal detection. The buzzer contacts are guarded from burning by capacitor *C*, and two dry cells furnish a d. c. supply of 3V.

This instrument can measure resistances independent of the resistance of the auxiliary earth electrodes.

To take an earth resistance measurement with the ИЗШ-52 (IZSh-52) tester, the lead from the earthing electrode or system under test is connected to terminal *1*, and terminals *2* and *3* to two auxiliary earth electrodes. The supply is switched on and the left-hand rheostat knob (marked "Coarse") is rotated until the note in the telephone reaches the lowest volume. Then the right-hand rheostat knob (marked

"Fine") is rotated until the note in the telephone is reduced as much as possible or ceases altogether. The earth resistance will be the sum of the two rheostat readings. When testing underground earthing circuits which have a resistance of less than 10 ohms, only the right-hand rheostat is employed, the left-hand rheostat being set to zero.
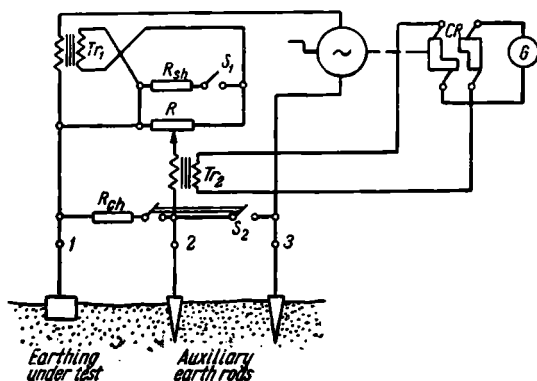
Fig. 258. Connections for measuring earth resistance with M1103 earth tester

The circuit of the M1103 intrinsically-safe earth tester is shown in Fig. 258.

It has two ranges: 1-10 and 5-50 ohms. In this instrument supply is provided by cranking its a.c. generator. The galvanometer G is connected across a commutating rectifier CR. The overall dimensions of the instrument are $232 \times 210 \times 166$ mm and its weight is 6 kg.

Measuring with this instrument is done as follows:

(1) The instrument is set truly horizontal and its cover is removed;

(2) Before the instrument is connected up, it is checked for proper operation. For this, with the earthing still disconnected, selector switch S_1 is set in the $\times 1$ position with changeover switch S_2 in its "Check" position;

(3) The instrument checked, the earthing under test and the auxiliary electrodes are connected to the instrument as shown in Fig. 258;

(4) By means of the zero-adjuster knob the galvanometer is set to zero.

(5) With the selector switch S_1 set in the $\times 1$ or $\times 5$ position (depending on the expected value of resistance), the changeover switch S_2 is thrown into the "Measure" position;

(6) The knob of the slide-wire rheostat R is rotated and the generator is cranked at 120 rpm until the galvanometer pointer returns to zero;

(7) The earth resistance will be the indication of the slide-wire knob multiplied by the factor against the selector switch S_1.

C h a p t e r   XIII

FUNDAMENTALS OF AUTOMATIC
AND SUPERVISORY CONTROL

13-1. General

Automatic control and telemechanics make up that field of engineering which develops and employs methods for control of industrial processes and systems. Automatic control is defined as control within a limited area (up to several hundred metres across), while supervisory remote control applies to control over distances running into hundreds of kilometres and requiring special apparatus.

The basis or starting point for automation is process mechanization, without which automation would be entirely impossible.

Automation obviates manual labour, and the worker has only to adjust and supervise operation of automatic plant.

Automation should be distinguished from ordinary remote control which implies operation of machines and installations by manual manipulation of control apparatus at some distance from the controlled plant. In coal mining, automation and remote control have found a broad field of application in underground transport, surface plant, pumping, ventilation, etc.

Any automatic or supervisory-remote control system consists of fundamental functional units, such as transducers (also known as transmitters or sensing elements), amplifiers, relays, selector switches, actuators, etc.

Transducer is the general name given to devices which convert the quantity being measured or controlled into some other quantity more convenient for acting on the control device. Transducers in automatic control systems most commonly convert some non-electrical quantity (force, liquid level, dimension, temperature, pressure, etc.) into a proportional electrical quantity. Fig. 259 shows a diagram of a wire or resistance strain gauge used to measure deformations in machine parts.

In it, a length of fine constantan wire (under 0.05 mm in diameter) is cemented in zig-zag form to a strip of paper 5 to 25 mm wide. This combination of paper and wire is bonded to the part under test and the gauge is connected to the supply and a measuring instrument. The elongation or compression of the part in the direction along which the wire has been arranged will then change its resistance and, consequently, the current in the measuring instrument. Elongation
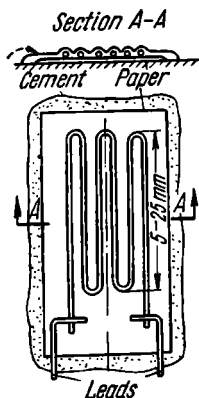
Fig. 259. Elements of a wire strain gauge
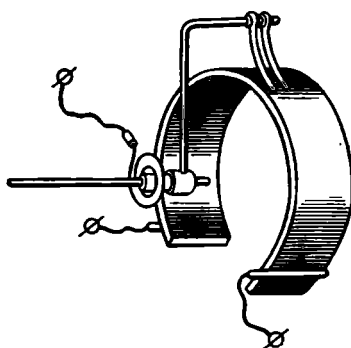
Fig. 260. Elements of a potentiometer type transmitter

stretches the wire and reduces its diameter; compression makes it shorter and thicker. The resultant changes in resistance give a measure of the deformation. In practice, the rosette type of gauge is often used when strains in several directions must be measured.

Fig. 260 shows a potentiometer-type transmitter, an ordinary rheostat with a slider contact. Such a transmitter can be adapted for obtaining indications of the level in water reservoirs by mechanically connecting its shaft through a system of levers to a float riding on the water surface. For level indication, the potentiometer is connected to a supply source and an indicating instrument with a scale calibrated in metres of liquid level. As the level of the liquid varies, the resistance in the measuring circuit is changed to produce a proportional increase or decrease in current in the indicating instrument.

Linear displacement is measured or indicated by means of induction-type transducers. A rotating-vane induction transducer (Fig. 261) consists of two a.c.-fed coils L_1 and L_2 fitted on steel cores between which a rotating vane is placed. When the vane is in its middle position, the air gaps on both sides are equal, and so is the inductive reactance of both coils. Displacement of the vane, say, to the right, will reduce the air gap on the right side, increase it on the

other. As a result, the inductive reactance of the right-hand coil will increase and that of the left-hand coil will drop* to produce a proportional change in the readings of the connected instruments.

Fig. 262 shows the essential elements of a capacitance transducer. It is an air condenser with a moving plate *1* and fixed plates *2* and *3*. If the moving plate is shifted, say upward, the capacitance of plates *1* and *2* rises while that of plates *1* and *3* decreases. By means of a suitable measuring circuit this change is detected and used for indication and control.
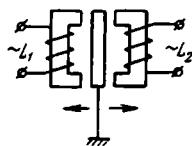
Fig. 261. Schematic diagram of an induction transducer
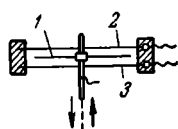
Fig. 262. Schematic diagram of a capacitance transducer
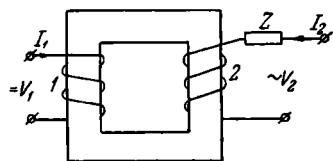
Fig. 263. Elementary magnetic amplifier

Induction-type transducers may also be used to measure speeds of rotation. Such a transducer will be designed as a small permanent-magnet machine (a tachogenerator) in which the terminal voltage is proportional to the speed at which its armature rotates. The latter is coupled in some manner to the shaft of the machine the speed of which must be measured.

Particular types of transducers such as thermocouple pyrometers are used for temperature measurement. The pyrometer comprises a thermocouple connected to a moving-coil permanent-magnet instrument. When the hot junction of the thermocouple is heated, the latter develops a thermo-emf dependent on the difference, in temperature between the hot and cold junctions of the couple. The indicating instrument has its scale directly calibrated in degrees. Pairs of materials used for thermocouples may be such as iron-copel (for up to 600° C) and chromel-alumel (for up to 1000 °C).

The second important element in automatic control system is the **amplifier**. This is a device which receives the output signal from a transducer and builds it up (amplifies) without any change in physical nature.

The need for an amplifier arises from the fact that the outputs from transducers are far too low in power for operating other elements of

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* When the air gap at the right-hand side is reduced, the number of lines of magnetic flux passing through the right-hand coil core increases. Since the magnitude of the flux threading the coil determines its inductive reactance, the latter also becomes larger.

a control system. The amplifiers used in automatic control systems may be of electrical and non-electrical types. The first include electronic, magnetic and rotary amplifiers; the second hydraulic, pneumatic and mechanical amplifiers.

The principle of operation of the electronic amplifier is discussed in Chapter Six.

Fig. 263 shows an elementary magnetic amplifier consisting of a steel core with two coils on it making up a saturable reactor. A d.c. voltage V_1 is applied to coil 1, while an a.c. voltage V_2 is applied to coil 2 with the load Z in series with it. By feeding coil 1 with direct current, the core is magnetized to saturation, the inductive reactance
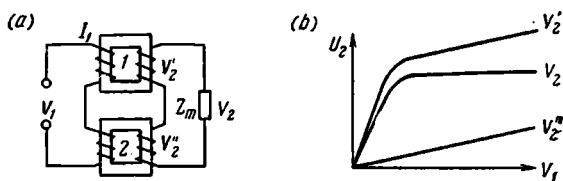
Fig. 264. Principle of operation of saturable-reactor
voltage stabilizer
(a) circuit; (b) magnetization curves

of coil 2 drops off, and current I_2 rises at constant V_2. When the core is operated at flux densities near saturation, even a slight change in the d.c. current in coil 1 will produce a large change in current I_2. This is the principle on which magnetic amplifiers work. By applying a very slightly varying d.c. voltage from a transducer to coil 1, greater changes are caused to occur in the a.c. current flowing through coil 2 connected in the circuit of a control device.

In its simplest form the rotary amplifier is a d.c. generator driven by an auxiliary motor. The terminal voltage of this generator will be proportional to its field excitation; its output voltage can therefore be controlled by varying its small field current. Its amplification factor is the ratio of the output power to the power fed to the field winding and may be anywhere between 80 and 100. Producing high power levels, the generator can drive large d.c. motors such as used in actuators.

Automatic control systems require power supply at very closely maintained voltages. The latter are held constant by devices called voltage stabilizers. An elementary saturable-reactor voltage stabilizer (Fig. 264) consists of reactors 1 and 2 which have their primary windings connected in series and supplied from the circuit with an unstable voltage V_1. The voltage V_2 which must be maintained constant across load Z_m is supplied by the secondary windings connected in opposition.

Due to design, reactor *1*, when energized with rated supply voltage V_1, will operate in the saturated region while reactor *2* will remain unsaturated. As the secondaries are connected in opposition, the voltage applied to the load will be equal to the difference between voltages V_2' and V_2''.

As voltage V_1 rises from zero to a maximum, secondary voltage V_2' rises at a high rate in the beginning but slows down as the reactor saturates (see voltage curve V_2'); voltage V_2'' at the same time rises
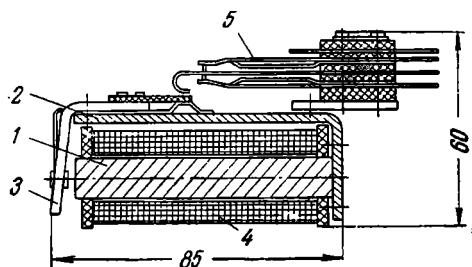
Fig. 265. Features of electromagnetic
attracted-armature relay:
1—core; *2*—magnetic body; *3*—armature; *4*—relay coil; *5*—contacts

directly proportional to V_1. The difference between these two voltages is the output voltage V_2. By designing the reactors so that the secondary voltage curves V_2' and V_2'' have the same slope relative to the horizontal axis, the output voltage V_2 can be maintained stable against wide changes in the supply voltage V_1.

Automatic control widely uses electric **relays,** or devices in which contacts make or break in response to variations in the conditions of one electric circuit and thereby affect the operation of other devices in the same or another electric circuit. By function, relays are of two groups: protective and control. The first protect electrical circuits and cause them to be disconnected from the supply when a disturbance in normal operation arises (short circuit, overload, etc.); the second switch circuits for purposes of control of production processes and equipment.

As to operating principle, relays, in general, may be electromagnetic, thermal, induction, mechanical, electronic, etc.

Fig. 265 shows the construction of a common electromagnetic attracted-armature relay very frequently found in automatic control systems. The magnetic circuit consists of a core, case and moving armature. As soon as the relay coil is energized, the armature turns to make or break contacts in the contact set. When the coil is de-

energized, the armature is returned to its initial position by a spring or by gravity.
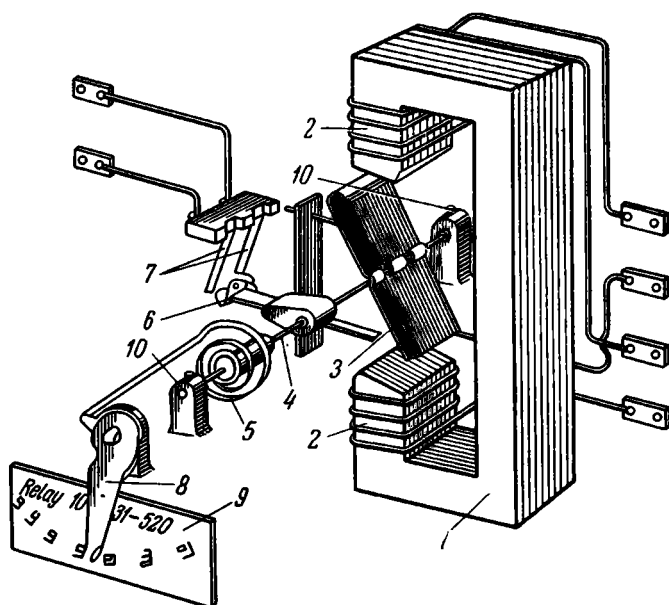
Fig. 266 is a diagrammatic representation of an ЭТ-520 (ET-520) electromagnetic overcurrent relay used for protection. In this relay core *1* carries two windings *2* on its poles and has a steel armature *3* mounted on spindle *4* placed between the poles. The spindle can turn in bearings *10*, carries the contact bridge *6*, and has one end of spiral spring *5* attached to it. The other end of the spring is connected to adjusting lever *8*. Varying the tension of the spring changes the current at which the relay will operate, the current settings being indicated on scale *9*.

When the current flowing through the relay coils exceeds the set value, the armature is attracted, and bridge *6* closes contacts *7*. Such relays are usually employed for overcurrent protection, i.e., for disconnecting a circuit if its load current becomes too great. Their coils are wound with a small number of turns and are connected in series.

Electromagnetic voltage relays which respond to the voltage in a given circuit do not in principle differ from the current relay described above. Such relays are designed to operate either when a set voltage
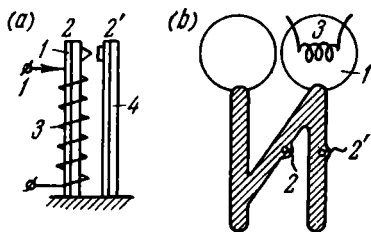
Fig. 267. Elements of thermal overload relays:
a—bimetal type; *b*—gas expansion type
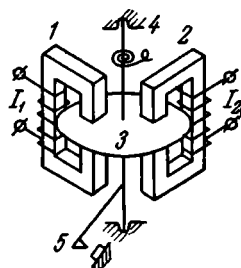
Fig. 268. Elements of an induction relay

is exceeded, or when the voltage falls too low or disappears altogether. In the first case they are called overvoltage, in the second case undervoltage or no-voltage relays. The coils in voltage relays contain a large number of turns and are connected in parallel with the circuit. In an overvoltage relay the armature is attracted and operated to open the circuit when the set voltage is exceeded; in an undervoltage or no-voltage relay the armature is maintained attracted and the contacts remain closed as long as the voltage is of normal value. On an impermissible drop in the voltage or its failure, the armature drops out, opens the relay contacts, and thereby opens the circuit protected by the relay.

Fig. 267*a* shows the essential elements of a bimetal-strip **thermal overload relay**. Bimetal strip *1* consists of two dissimilar metals with different coefficients of linear expansion which are bonded to each other. It carries a contact *2*, and is placed inside a heater coil *3* through which the circuit current is passed. When an overload causes sufficient heating of coil *3*, the bimetal strip bends to one side due to the difference in expansion of its metal layers and closes contacts *2* and *2'*. To reduce the effect of ambient temperature upon the relay, contact *2'* is also mounted on a bimetal strip *4*.

Fig. 267*b* illustrates the operating principle of a thermal relay based on the expansion of a gas.

Glass bulb *1* is filled with hydrogen which can be raised in temperature by the heater coil *3*. On heating due to the current passed through the coil, the hydrogen expands. This expansion displaces some of the mercury in the left-hand well and thereby opens the contacts *2* and *2'*.

An **induction type relay** consists essentially of a rotatable aluminium disc *3* placed in the magnetic fields of two electromagnets *1* and *2* (Fig. 268).

The electromagnets, fed with a.c. current, induce currents in the disc which then react magnetically with the alternating magnetic

fluxes of cores *1* and *2* and make the disc rotate until it closes contacts *5*. The restoring force is supplied by a spiral spring *4*. This type of relay is extensively used for protection.

Another type of relay finding general application in protection and automatic control circuits is the time delay relay, a device which, when energized, will open or close its contacts after a preset lapse of time from the moment it was energized. The type usually found in
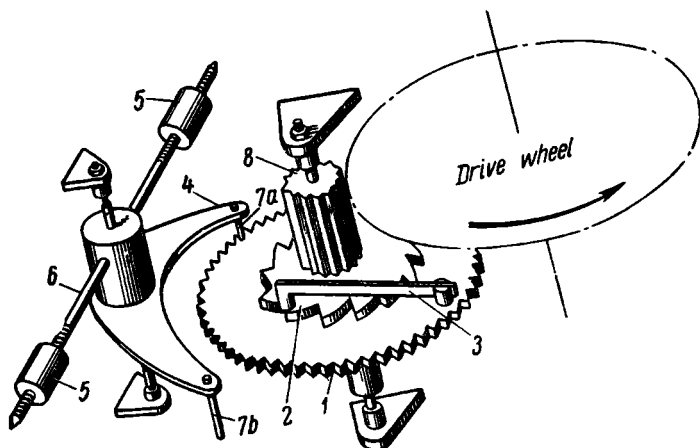
Fig. 269. Clockwork movement of a time delay relay

protective circuits is the clockwork time delay relay, while the motor and electronic types (also called timers) are used in automatic control.

Fig. 269 shows the key parts of an ЭВ-180 (EV-180) clockwork time delay relay. The movement is depicted at the moment when pin *7a* of anchor escapement pallet *4* has engaged the teeth of scape wheel *1* and stopped it to rebound. This throws it out of engagement, so that the scape wheel and the movement connected to it continue to operate until pin *7b* on the other pallet arm engages the teeth and stops the wheel. Since pin *7b* likewise receives a recoil impulse, it makes pallet *4* turn backwards. Thus the scape wheel rotates intermittently.

The drive wheel incorporated in the movement to initiate motion is seated on the same shaft which operates the relay contacts. When the relay is energized, its rate of rotation will depend on the position of adjustable pendulum weights *5* on crossarm *6*. As they are moved further from the fulcrum, a greater time delay is obtained. For quick resetting of the contact operating mechanism, the drive gear is coupled to scape wheel *1* through a pawl and ratchet. The pinion *8* receiv-

ing motion from the drive wheel is fixed relative to ratchet wheel *2* and, since the end of pawl spring *3* is held in engagement with the ratchet teeth and spring *3* is attached to the scape wheel, scape wheel *1* turns together with pinion *8*. The mechanism is returned to its initial position by turning the drive wheel in the opposite direction. In this direction the ratchet teeth easily slide past the end of pawl spring *3*, leaving scape wheel *1* at a standstill and the mechanism free to return quickly to its initial position.

Fig. 270. Schematic diagram of a motor-type timer

Fig. 270 gives the schematic diagram of a synchronous motor timer providing a time range from a split minute up to several hours. When its starting contact *K* is closed, it energizes a miniature synchronous motor *M* coupled to a gear train and an engaging electromagnet *EM*. The latter brings gear wheels Z_1 and Z_2 into mesh and thereby makes cam *C* turn in the direction indicated by the arrow. As the notch of the cam, comes under the end of lever *L*, the latter turns under the action of spring S_3 and thereby opens contacts *1* and *2*, and closes contacts *3* and *4*. When contacts *1* and *2* open, they stop the motor and cam. The cam remains at a standstill as long as starting contact *K* is held closed. Opening contact *K* causes the engaging electromagnet to release its armature and allow spring S_1 to pull gear wheels Z_1 and Z_2 out of mesh, and spring S_2 to return cam *C* for a new start from stop *A*. Contacts *3* and *4* are connected in the circuit controlled by the relay.

The valve-type static timer (or electronic timer) is of a class of timing relays utilizing a capacitance and a thermionic valve*. Relays of this type provide ranges from a few hundredths of a second to about 5 minutes. The elementary circuit of such a relay is shown in Fig. 271. Initially, a switch or contact K is closed and capacitor C
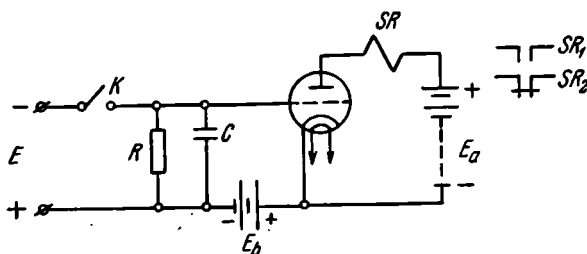
Fig. 271. Electronic time delay circut

is charged to a voltage E which maintains the valve grid at negative potential $E+E_b$ (E_b being the potential grid bias battery and E being the supply voltage of such polarity that they are additive).

The valve is therefore blocked and no current flows in either the anode or grid circuit. From the moment contact K is opened, capacitor C discharges through resistor R at a rate dependent on the value of the resistor until the negative potential of the grid within a definite interval of time drops to a point at which the valve becomes conductive and passes anode current through relay SR to operate it and change over its contacts SR_1 and SR_2. Reclosing contact K makes the relay ready for a new cycle because the high negative potential again applied to the valve grid blocks the valve, and causes the relay to reset.

Automatic control and monitoring of processes occurring within closed vessels uses electronic relays based on a gamma-ray source.

If a small piece of a radioactive element is placed in a lead chamber with a window and a photographic plate is placed opposite the window (Fig. 272), the rays escaping through the window will produce a single black spot on the plate on exposure. If the rays are passed between two electrically charged plates, they divide into three beams which will leave three spots on the exposed photographic plate. These beams, differing both in nature and physical properties, have received the names: alpha, beta and gamma. Alpha rays are positive and deflect towards negatively charged plates; they consist of charged particles (identical with the nucleus of a helium atom). They are

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* The principles of thermionic valves were discussed in Chapter Six.

easily retarded by a layer of air several centimetres thick or thin plates of aluminium. Beta rays deflect towards positively charged plates and consist of electrons capable of penetrating several metres of air or a plate of aluminium some millimetres thick. Gamma rays do not deflect either way. They are similar to light rays, that is, are
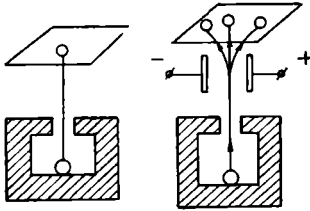
Fig. 272. Nuclear radiations
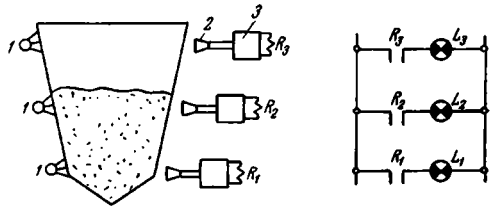
Fig. 273. Diagram showing operation of a radioactive level indicating system

electromagnetic waves of very high frequency and similar to X-rays in properties. They can penetrate metals tens of centimetres thick and are utilized in industry for monitoring a great many processes.

Fig. 273 shows diagrammatically operation of a level indicating system based on so-called gamma-ray relays each comprising a radiation source *1*, radiation detector *2*, amplifier *3* and an electromagnetic relay *R*. Where the hopper ·is empty, only its walls separate source *1* and detector *2*. The radiation reaching the detector produces a signal which blocks the amplifier, leaving relay *R* deenergized. However, if the material fills the space between the walls, some of the radiation is absorbed, and the signal produced by the remainder reaching the detector will be unable to keep the amplifier blocked, the relay will be energized. The relay closes its

Fig. 274. Parts of a telephone-type selector

contact in the circuit of the respective signal lamp *L* to indicate the level in the hopper.

The use of various relays for protection and automatic control has been discussed earlier in Chapter Eight.

A further element of automatic control systems is some form of selector by which a given circuit may be successively switched to a series of other circuits. The selector is used where it is necessary to control or monitor several units of equipment by means of one device.

In most cases electromagnetic stepping relays called selectors serve this purpose. They have wipers which move in steps for making successive connections to the necessary circuits. Fig. 274 shows the parts of a telephone-type selector. Its electromagnet, when energized, attracts the armature to move the pawl by one tooth of the ratchet wheel fixed on the same spindle as the wipers. De-energizing the electromagnet allows a spring to return the armature to its former position, while pawl C_1 turns the ratchet wheel and the wipers one step further. A second pawl C_2 locks the wheel and wipers on each step. The wipers sweep over a bank of fixed contacts *1*, *2*, etc. The electromagnet in this device incorporates an interrupter *1* which opens its contact each time the electromagnet is energized.

The electromagnet will therefore be switched on and off periodically (like an electric bell), stepping the wipers across the bank of contacts.

Last but not least, automatic and supervisory-remote control systems have final control elements, which may be motors, actuators, or other devices. Most often they are motors of fractional rating (miniature motors) which are easy to reverse, accelerate, or slow down within a wide rpm range. D.c. series- and shunt-wound motors and squirrel-cage two-phase induction motors are used for this purpose.

13-2. Principles of Automatic Control

An automatic control system will maintain a machine or process in operation without any assistance on the part of an operator continuously and at required accuracy in accordance with the conditions for which it has been set. Such a system, in replacing a human operator, must perform the following operations:

(1) Measure the controlled variable;

(2) Compare the measured value of the controlled variable against the set point so as to determine the deviation from the set point, or the error;

(3) Act upon the process or plant so as to minimize the error.

A typical system will incorporate the following elements:

(1) A measuring element which determines the actual magnitude of the variable most representative of the desired process or plant condition and sends out a signal representative of this magnitude to a controller;

(2) A controller which compares the actual magnitude of the variable against the set point or reference value;

(3) A final control element which acts upon the process or plant so as to minimize the error.

Several automatic systems have been developed recently for the control of mining machines. One of them controls the load of the main and loader motors of the Donbass cutter-loader. The desired load

current of the main motor is set by the operator with a rheostat. Any deviation from the set point produces an error signal at the output of a magnetic amplifier which drives the actuator so that the latter, through a transmission, varies the speed of advance of the cutter-loader accordingly, reducing the speed of advance with increasing load and stopping it up in the opposite case. The magnetic amplifier also responds to any overload on the loader motor, slowing down the cutter-loader advance until it has "worked up" excess coal and its motor current drops to normal.

Another automatic control system is intended to steer heading or tunneling machines. The direction sensing element is a pendulum whose deflection, when the machine deviates from its set course, is detected by air-gap induction pickups. They feed an error signal through a magnetic amplifier to a computer. The latter also receives signals of distance in advance, attitude of the cutter head at the face, etc. The computer evaluates the signals and operates the hydraulic controls of the cutter-head mechanism through an actuator.

The principles of automatic control also apply to the automatic production lines (transfer machines), i.e., systems of machines which automatically perform a predetermined sequence of operations on the material or workpiece. The workpiece or material is moved from operation to operation and automatically indexed in each work station by automatic transfer devices until all the necessary operations have been performed on it. In the U.S.S.R. there are automatic lines for machining truck cylinder blocks, castings, tubes and pressed products, and high-speed welding of pipes. Similar equipment is in use in the light and food industries.

13-3. Supervisory Control Systems

Supervisory control systems operate over great distances measured in tens and hundreds of kilometres. As compared to automatic control systems, they additionally include communication channels, and communication transmitting and receiving equipment. The function of the transmitters is to convert any controlled variable (for example pressure, force, etc.) into a quantity convenient for transmission over a communication link. At the other end of the link the signal is fed into a receiver to be again converted to a form suitable for acting on a final control element.

Supervisory control systems are shown in block-diagram form in Fig. 275.

Fig. 275*a* shows a two-way supervisory control system. It differs from an automatic control system in that it includes a transmitter and receiver at the controlled plant *1*, two communication links *4*, and a transmitter-receiver unit *5* at the controlling point.

In the case of a telemetering system (Fig. 275*b*) the controlled quantity is converted into a signal by a sensing element *2*, which is fed to transmitter *3*. The latter relays it over the communication
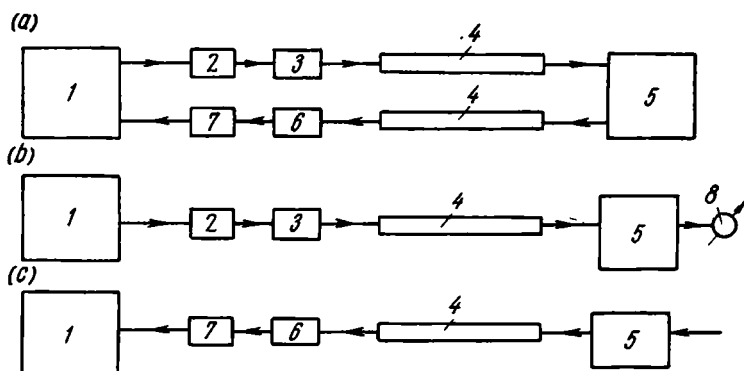
Fig. 275. Supervisory control systems:

a—two-way supervisory control system; *b*—telemetering system; *c*—single-way supervisory control system; *1*—controlled plant; *2*—sensing element; *3*—transducer; *4*—communication link; *5*—receiver-transmitter; *6*—receiver; *7*—final control element; *8*—indicating-recording instrument
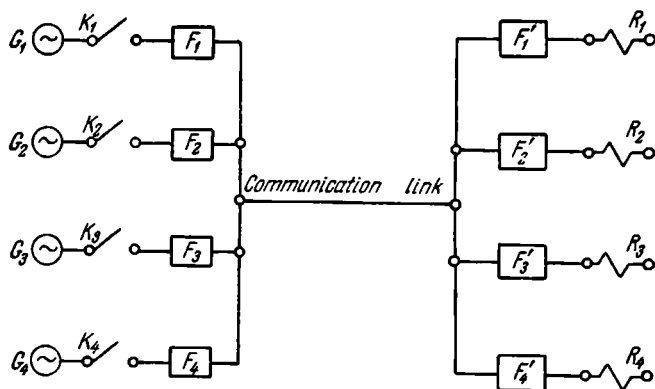
Fig. 276. Single communication link used for control of four plants

link *4* to the receiver *5* for indication or recording by a measuring instrument *8*.

In the remote control system shown at *c* in Fig. 275, the command sent from the operator's desk goes through a transmitter *5* and communication link *4* to reach a receiver and be relayed to the final control element to produce the desired effect in the controlled plant *1*.

With supervisory control, it is possible to operate a great many installations from a single central control point. It is widely employed for the operation of power systems from the dispatcher's office situated very far away from the numerous generating and switching stations and substations.
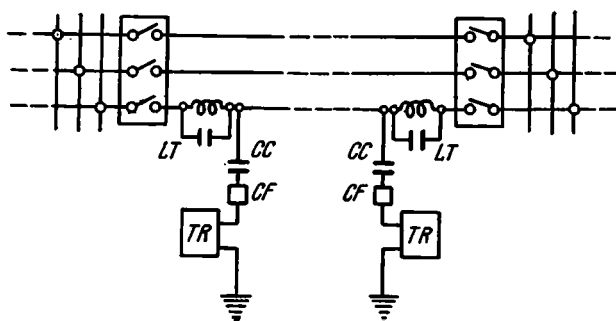
Fig. 277. Use of power-line carrier for power system communication

The essential elements of each supervisory control system are transmitters, receivers and communication links. When the controlled plant units are many, provision of several communication links would involve considerable capital outlays. Instead, use is made of one communication link which can transmit simultaneously several different signals (i.e., it can provide several channels or bands of frequencies).

Fig. 276 illustrates the scheme whereby one communication link serves four controlled plants. At the transmitting-end four generators G_1, G_2, G_3 and G_4 are installed to supply currents of different frequencies to the communication link through filters F_1, F_2, F_3 and F_4. The four filters F_1', F_2', F_3' and F_4' at the receiving end feed relays R_1, R_2, R_3 and R_4, respectively.

When, say, the key K_3 is closed, the generator G_3 sends a current of definite frequency into the communication link. At the receiving end only filter F_3' will pass this frequency, and, of the four relays, only the relay R_3 will operate. The other filters will block this frequency to prevent them from reaching the other relays.

Interconnected power systems are widely employing what is known as power-line carrier communication. In such communication systems, the high-frequency intelligence modulates the high-voltage current and there is no need for separate communication links (Fig. 277). Signal transmission over high-voltage lines is accomplished at frequencies from 30 to 500 kc/s. The transmitter-receiver TR in this system is connected to one of the three line conductors through a high-voltage coupling capacitor CC and a coupling filter CF. The

coupling capacitor easily feeds the high-frequency current into and from the line conductor but offers an enormous resistance to the flow of 50 c/s high-voltage current.

The line trap *LT* consisting of an inductance coil and a capacitor prevents the high-frequency current from flowing further into the power circuit at the ends of the line conductor; it has a very small
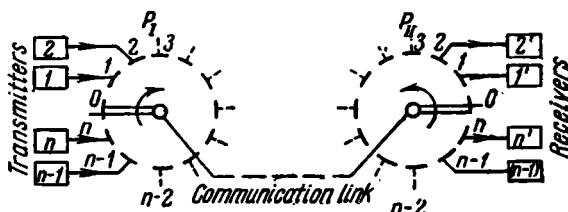
Fig. 278. Operation of selectors

resistance for the industrial frequency current. Several messages can be sent simultaneously over the line conductor (i.e., several channels can be obtained) by using several frequencies from different transmitter-receivers to modulate the power-line current.

Fig. 278 shows a circuit arrangement by which several signals can be transmitted over one link by placing selectors P_I and P_{II} at the transmitting and receiving stations. The contacts of selector P_I are connected to a set of signal transmitters *1, 2*, etc., while a set of respective receivers are connected to selector P_{II}. Since the wipers in the selectors run in synchronism, they simultaneously pass over like contacts to complete the path between them. The fact that there is permanent connection between the transmitters and receivers, with the wipers constantly rotating, does not put this system at a disadvantage because the wipers revolve at a very high speed and the time required for switching is extremely small.

Chapter XIV

POWER FACTOR
AND POWER ECONOMICS

In operation induction motors and transformers draw both active and reactive power or energy from the supply. The active power goes to perform useful work and cover the losses in the motors, transformers and circuits. The reactive power plays no part in performing useful work, but it creates the magnetic fields in motors and transformers. Thus, two power components, active P and reactive Q, are present in the windings of motors and transformers and in the conductors of the supply circuits. The square root of the sum of their squares is termed the apparent power S.

For three-phase systems the active, reactive and apparent power can be calculated by the following formulas:

$$P = 1.73VI \cos\varphi, \text{ kW;}$$
$$Q = 1.73VI \sin\varphi, \text{ kVAr;}$$
$$S = 1.73VI = \sqrt{P^2 + Q^2}, \text{ kVA.}$$

The ratio of the active power to the apparent power is called the power factor, i.e.,

$$P.F. = \frac{P}{S} = \frac{P}{\sqrt{P^2 + Q^2}}.$$

The power factor, which is numerically equal to the cosine of the phase angle, shows what part of the apparent power provides the active power. At a power factor ($\cos\varphi$) equal to unity, the apparent power is entirely active.

For practical purposes, the power factor of a mine can be calculated from the readings of the active and reactive meters installed in the surface substation. It is usual to calculate what is termed the mean

weighted power factor from the monthly consumptions. If, for example, the active meter shows that the monthly consumption of active energy has been $W=800,000$ kWh and the reactive energy meter gives a reactive energy $W_r=500,000$ kVArh, the mean weighted power factor will be

$$P.F. = \frac{W}{\sqrt{W^2 + W_r^2}} = \frac{800,000}{\sqrt{(800,000)^2 + (500,000)^2}} = 0.85.$$

Within the districts, the main consumers of reactive power are the induction motors; they therefore determine the power factor at which the district circuit operates. As for the motors themselves, their power factors depend on the loads they carry. Lowering a load on a motor reduces its power factor. If, for example, a certain motor has a power factor of 0.85 at full load, its power factor will drop to 0.74 when it is run at half-load; at quarter load the power factor drops to 0.55. The power factor is particularly low at no-load (0.15 to 0.25).

The less the power factors of industrial consumers the greater the power ratings required of the generators in the stations and of the transformers in the substations. Consider a generator at a power station, rated for $S=25,000$ kVA and supplying the load at 0.9 power factor. In this case the generator can provide an active power output of $P_1=S \cos \varphi_1=25,000 \times 0.9=22,500$ kW. If, as a result of underloading, the load power factor decreases to 0.6, the generator can only have an active output of $P_2=S \cos \varphi_2=25,000 \times 0.6=15,000$ kW. To cover the loss in station capacity $P=P_1-P_2=22,500-15,000= =7500$ kW, it becomes necessary to install an additional generator rated for $7\,500 \div 0.6=12,500$ kVA.

Now consider a mine with an active power load $P=1\,800$ kW and a power factor of 0.9. The total capacity of the transformers to be installed in the surface substation will then be

$$S = \frac{P}{\cos \varphi_1} = \frac{1800}{0.9} = 2000 \text{ kVA}.$$

To meet the power requirement we would install three transformers, each of 750 kVA capacity. However, if the power factor drops to 0.6, the substation would then require a fourth transformer of 750 kVA rating because

$$S_2 = \frac{P}{\cos \varphi_2} = \frac{1800}{0.6} = 3000 \text{ kVA}.$$

Low power factor leads to the need of increasing the sizes of wire and cable conductors, and to loss of power and drop in voltage in generator and transformer windings and also in power lines.

Assume that a cable is carrying an active current $I_a=200$A at a power factor of 0.92.

The full current of the cable will then be

$$I_1 = \frac{I_a}{\cos \varphi_1} = \frac{200}{0.92} = 217 \text{ A.}$$

Such a current makes it necessary to install a 70 mm² cable with a current-carrying capacity of 245A at 6 kV. If the power factor decreases to 0.6, the full current will rise to

$$I_2 = \frac{I_a}{\cos \varphi_2} = \frac{200}{0.6} = 333 \text{ A,}$$

a current which makes us take a cable of up to 120 mm² size with a current-carrying capacity of 340A.

The power factor can be improved through organizational and technical measures. Those in the first group may be:

(a) maintenance of high load on induction motors and transformers;

(b) elimination of idle running of induction motors by means of special limiting devices;

(c) use of synchronous induction motors which, when over-excited, will feed reactive power into the circuit for supply of other loads.

Another measure in this group is practiced by Power Authorities whose rates for power consumed are scheduled according to the consumer's power factor.

At coal mines synchronous motors are used to drive the up-cast (main) ventilating fans and the compressors. The synchronous motor, like a slip-ring or a squirrel-cage motor, has a stator with a three-phase winding. Its rotor, however, carries a field winding excited with direct current supplied by a small d.c. generator known as the exciter. The latter is usually seated directly on the motor shaft. By nature of its design and excitation, the rotor becomes an electromagnet and, when the stator winding is connected to a three-phase supply and produces a revolving magnetic field, the latter links with the rotor field so as to make the rotor turn in the same direction. The speed of the rotor is exactly the same as the stator field, i.e., is equal to synchronous speed. This fact has led to the name synchronous motor. Increasing the exciting current in the field winding above that needed for normal motor operation will cause the synchronous motor to return excessive reactive power to the supply circuit for use by induction motors. Such a motor designed only for power factor improvement is known as the synchronous condenser.

The principle technical measure for improving the power factor is to connect high-voltage static capacitors across the line. A high-voltage capacitor consists of conducting plates in the form of aluminium foil wound so that it is separated by a thin layer of oil-impregnated paper. These plate assemblies are placed in metal cases subsequently filled with oil. The covers sealing the cases carry bushings

fitted with terminals for connecting the plates to the power circuit. When a static capacitor is connected across the circuit, it acts as if it were a local generator of reactive power, replacing to a given extent a station generator. Such capacitors are arranged in banks and are installed in 6 kV surface substations. The induction motors in the mine and the transformers in the substation can thus receive their necessary reactive power from the capacitors which thereby free the station generators, power region transformers and transmission lines of this load.

The mine electrician should never allow the equipment to run idle as this will lower the power factor and waste electric energy. When picks are being replaced on a coalcutter or a cutter-loader, or when props are being moved up, the face and gate conveyor lines should be shut down to keep the conveyors from running idle.

District substation transformers should be selected so that they will be as fully loaded as possible.

Excessive transformer capacity means that the transformer will operate underloaded, that the power factor will be low and that more money will have to be paid for the consumed power.

On the other hand, insufficient transformer capacity in the substation will result in lowered voltages across the motors, making it necessary to keep their loads down and thus reducing the output of the equipment at the face. The same thing will occur if the sizes of the armoured and flexible cables are too small and therefore have large voltage drops.

Where the distance from district substation to the working faces is allowed to become great and the cables laid between them are of inadequate size, the drop in output of face machinery is accompanied by heavy losses in the distribution circuit.

Chapter XV

MINE TELEPHONY

Various types of telephone communication are in use in mines. A general mine telephone system usually connects the various scenes of mine work between themselves and with the supervisory staff.

At large coal mines the general telephone system is supplemented by a selective party line.

Sometimes local telephone circuits are provided between the onsetter, banksman and the winding engineman, or between cutter-loader operators and the loading point stations.

In addition to conventional telephone, use may sometimes be made of carrier telephony between the dispatcher and the electric locomotive drivers.

Underground Telephone Sets

Telephone sets for underground service must be of special design. In general, telephone sets may be of the common-battery, local-battery, and voice-operated types.

Fig. 279 shows the basic circuit of the ТАШ-53м (TASh-53m) common battery mining telephone set. These sets are intrinsically safe, which means that the sparks that can appear if the telephone cable is injured will not ignite the methane-air mixture because the sparks are weak. The circuit of this set has a capacitor C connected in parallel with the hook switch HS and also includes a neon tube NT.

In all other respects the ТАШ-53м (TASh-53m) set does not differ from a conventional common battery set. Its case is of silumin (an alloy of aluminium and silicon) with the cover sealed by a rubber gasket. The handset HS is connected to the set by a flexible cord

protected by a wire spiral. The call bell *CB* is mounted on the back wall of the set.

Common battery telephone sets draw their power from the storage battery installed at the telephone exchange. In the local battery system each telephone has built-in dry cells.
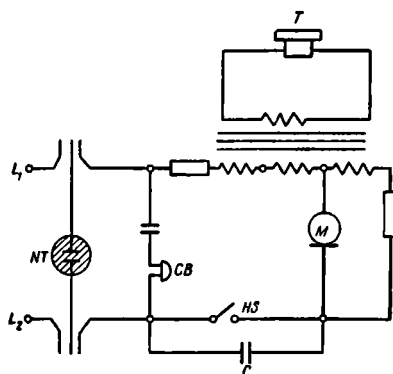
The local-battery МБ-ВЭФ (MB-VEF) mine telephone set (Fig. 280) has a microphone (transmitter) *M*, two receivers T_1 and T_2, transformer *Tr*, battery *B*, induction coil *IC*, and a call bell *CB*. The hook switch *HS*, when a receiver is lifted from the hook, connects the microphone to the battery, while coil switch *CS* connects induction coil *IC*.

The МБ-ВЭФ (MB-VEF) set has two cast-iron casings; one for the set, the other for the battery, both being joined by means of a steel frame. The case containing the set carries the transmitter on its front wall protected by a cover with holes in it. The a.c. type call bell is mounted on its top wall. This telephone set is of general-purpose protected mine design and its use is therefore limited.

Fig. 279. Circuit of ТАШ-53М (TASh-53m) telephone set
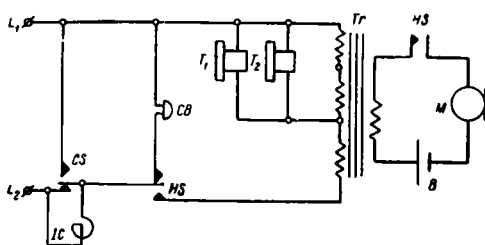
Fig. 280. Circuit of МБ-ВЭФ (MB-VEF) telephone set

Voice-operated sets require no power source because the microphone converts sound vibrations into the respective electrical signals. Voice-operated telephone sets are used by mine rescue crews.

Communication of the Cutter-loader Operator with a Loading Point

Communication between a cutter-loader and a loading point uses the same trailing cable by which the power is fed to the cutter-loader's motors. The three auxiliary cores in the cable used for this
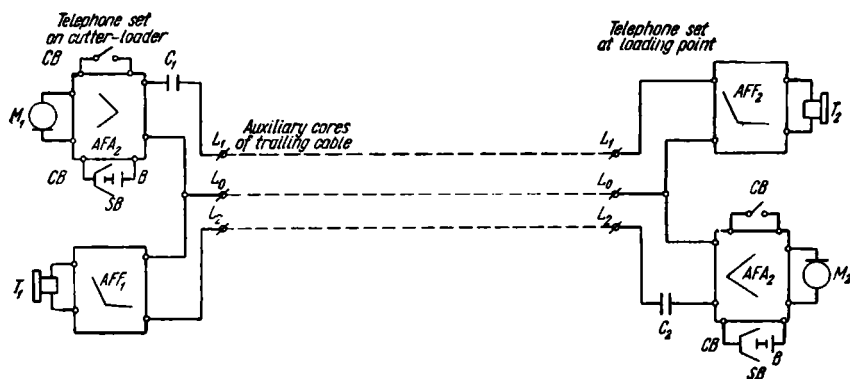
Fig. 281. Telephone circuit for communication between cutter-loader operator and loading point

purpose are connected to terminals L_1, L_2 and L_0 at the telephone sets (Fig. 281).

To call the loading point, the operator of the cutter-loader pushes call button *CB* on the telephone set. This sends a calling tone of 1200 c/s frequency to the set placed at the loading point. During a conversation, the speech buttons *SB* of each set must be held depressed. These sets incorporate audio-frequency crystal amplifiers *AFA* to amplify the speech and call currents reaching the receiver insets.

Each set uses four dry cells *B* with an overall voltage of 6V.

The microphone *M* incorporates an electromagnetic inset which is also used in the calling-tone and speech receiver *T*, fitted with a horn.

Any interference from the industrial frequency currents passing through the cable together with the communication currents is eliminated by connecting the receiving circuits of the sets through audio-frequency filters *AFF*. The latter are a combination of inductances and capacitors designed to pass currents only of the required frequency.

This telephone set is of intrinsically safe design and can therefore be used in gassy and dusty mines. The weight of the set, complete with power supplies, is 8 kg.

High-frequency Carrier Communication Between an Electric Locomotive Operator and the Dispatcher over Trolley Wires

This form of communication uses the electric locomotive trolley wire as one conductor and the track rails and earth as the other.

To keep the d.c. power currents supplied to the motors from reaching the telephone sets, and at the same time prevent the high-frequency current sent to the telephone sets from being short-circuited by the motor windings, blocking filters are inserted in series with the locomotive motors.

Chapter XVI

SAFETY IN COAL MINES

The Mine Atmosphere

Air normally contains 79 per cent nitrogen, 20.96 per cent oxygen and 0.04 per cent carbon dioxide.

A mine atmosphere, however, is different.

According to Soviet safety regulations, the atmosphere where miners may work must not contain more than 0.5 per cent carbon dioxide.

Carbon dioxide is heavier than air and gathers at the floor. Its presence can be therefore detected by a flame safety lamp lowered to a point near the floor. If the content is high, the flame will go out.

The mine atmosphere may contain other gases; for example, methane (or firedamp, as it is called by miners), carbon monoxide, hydrogen sulphide, sulfur dioxide.

Methane is a colourless gas half as heavy as air and tends to gather at the roof. When its percentage is high in the atmosphere, it can lead to suffocation of men working at the headings because it replaces the oxygen in the air. Mixtures of methane and air are inflammable and will be exploded by an open flame or electric spark. At contents of from 5.5 to 14 per cent in air, the mixtures become very explosive. If the methane percentage rises to 14 per cent and higher, the mixture will not support combustion or explode. The greatest force of explosion occurs when the air contains 9.5 per cent of methane. In coal mines methane is given off at rates which vary from a continuous and slow bleed-off to sudden outbursts which persist only for a short time but discharge large volumes of the gas.

Mines in which firedamp is given off in even the smallest amounts are immediately classed as gassy and may be operated only as such. The use of open flame in them is prohibited.

The electrical equipment in coal mines must be kept under close and constant watch in order to prevent any possibility of arcing or sparking.

Should the methane content in the atmosphere of a working or entry face rise to 2 per cent, all work must be discontinued, the men withdrawn from the gassy workings and the electric power cut off until the percentage of methane is reduced to one per cent.
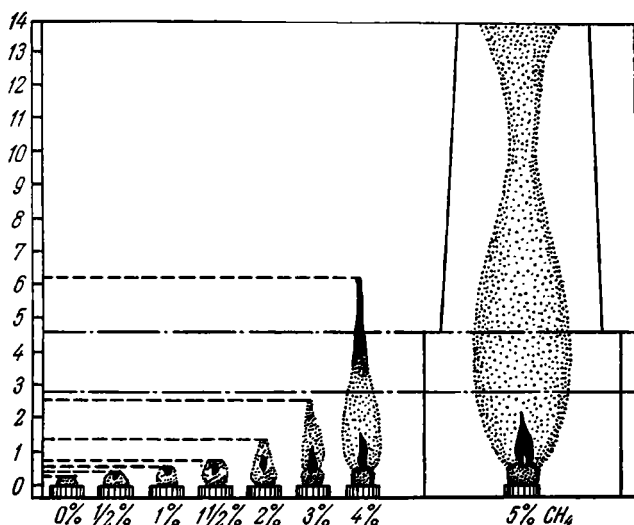
Fig. 282. Determination of methane content of mine atmosphere

No shot firing is allowed when the methane content at the face has reached one per cent.

The methane percentage in a mine atmosphere can be determined by means of a petrol safety lamp.

For testing, the lamp, with its flame burning at normal height, is slowly raised towards the roof. If the flame in the lamp begins to spire up and soot appears, the test should be discontinued since this is an indication of a methane content of over 4 per cent. The lamp should be carefully lowered, and the miner should go out into the fresh air stream.

Should the lamp flare up during the test with a full-size flame, lower the lamp slowly, and leave the face.

If in the first test with a full-size flame the flame does not rise, another test must be made with the flame reduced to about 2 mm. As in the first test, the lamp must be raised slowly.

If there is methane in the air, a pale-blue cap of burning gas forms above the flame. The greater the methane content in the air, the higher the flame (Fig. 282).

If the top of the "gas cap" reaches the middle of the lamp glass, this is an indication of a methane content of over 2 per cent. Discontinue the test, lower the lamp slowly, leave the working and switch off the electric power.

Coal Dust

Air-suspended coal dust coming from seams containing volatile matter is easily exploded, especially if the air contains methane.

The risk of a coal dust explosion can be reduced by means of stone dusting, stone barriers, and watering of the workings.

The inert dusts used for stone dusting must be dry, finely ground and free from inflammable material or sand.

Large quantities of coal dust are produced during the operation of coalcutters, cutter-loaders and conveyors. A great deal of coal dust is also produced at the car loading points. This is where provision of water sprayers is essential.

Modern coalcutters, coal and tunneling cutter-loaders are fitted with water spraying equipment to deal with this problem.

Ever wider use is being made of water infusion by which water is injected into the seams beforehand, thereby reducing dust formation.

Mine Fires

Finely-broken coal is sometimes left in the workings. When the coal is liable to self-ignition, it first gives up heat and then ignites. Therefore, a characteristic smell of burning is felt before a fire breaks out.

A fire can also be started by an open flame or spark. Any breakdown in the electrical equipment may be the cause of an underground fire, or methane and coal-dust explosion. Metal boxes filled with sand and dry-type (sand) fire extinguishers must therefore be provided at all places where electrical equipment is installed. All lubricating materials must be kept in metal boxes fitted with covers.

If a fire has broken out in some unit of electrical equipment, its power supply must be cut off immediately, the mine inspector notified, and all workmen near the equipment warned of the fire.

Throughout the coal mining areas in the U.S.S.R., rescue crews are continuously on the alert and, the instant a signal of fire or explosion is received, arrive as quickly as possible to take over the rescue work.

Mine Ventilation

The best way to cope with methane is to adequately ventilate the workings.

Ventilation should provide such a quantity of air that the main upcast shaft air will never contain more than 0.75 per cent of methane, while the content in the return airway of a district should not exceed 1 per cent.

The movement of the air stream is induced by the difference in pressure between the air in the mine and the air at the surface, a difference created by the ventilating fans exhausting the upcast air from the mine.

Proper distribution of the air to all the mine workings is achieved by putting in ventilation doors to shut off the flow of intake air to points where it is not needed and thus increase the air flow to the workings where it is required.

Electrical Safety Regulations for Underground Coal Mine Workings

Electrical machines, apparatus and transformers of only mine design are permitted for use in Soviet collieries.

Relevant Soviet standards divide them into general mine (marked PH on the enclosure), higher-reliability (PΠ) and flameproof (PB) types. Intrinsically-safe mining electrical equipment is marked with the letters PBИ.

In gaseous and dusty mine workings every unit of portable, transportable and self-propelled equipment (coalcutters, cutter-loaders, conveyor drives, district power distribution switchgear and gate-end boxes, etc.) must use certified flameproof mining electrical equipment. Such electrical apparatus have explosion-proof or flameproof enclosures designed for extra mechanical strength to withstand an internal pressure of 8 atm (gauge) within an enclosure volume of over 2 litres. Moreover, the shape, flanges and the gap at the joints of the enclosures are such that any flame due to an explosion within the enclosure will be adequately cooled and prevented from escaping into the surrounding atmosphere.

The width of joints and other mating faces in flameproof enclosures must be of the following size:

> 5 mm for enclosures with internal volumes up to 0.1 litre;
> 8 mm for enclosures with 0.1 to 0.5-litre internal volumes;
> 15 mm for enclosures with 0.5 to 2-litre internal volumes;
> 25 mm for enclosures with internal volumes over 2 litres.

All the mating surfaces of various flameproof enclosure parts, including the cable glands and sealing-ends, the surfaces of joints

and places where the shafts and control rods pass out must be machined to a smooth finish.

The enclosures of low-power electrical equipment for connection to a secondary supply voltage not over 133V and under 4 kVA in rating may have a gap not greater than 0.2 mm at any point between their fixed flat jointing surfaces and not have a diametral clearance greater than 0.3 mm at any point between their fixed cylindrical mating surfaces.

The enclosures of electrical equipment of greater power capacity intended for supply voltages higher than 133V and rated for over 4 kVA must be manufactured so that the gap between fixed flat jointing surfaces and the diametral clearance between fixed cylindrical mating parts will not exceed 0.1 mm in enclosures with free internal volumes of 0.5 to 10 litres, and will not exceed 0.2 mm when the free internal volume is over 10 litres.

Compliance with the above gap and clearance requirements is a matter of vital importance. If the cover of a controller on a coal-cutter is removed while underground and is then replaced, it may not be fully tight, due to the coal dust which settles on the joint surfaces, thereby increasing the gap. Because of this, there is the danger of the methane-and-air mixture in the controller being ignited as the contacts make and break. The resultant flame will be able to escape through the oversize gap into the surrounding atmosphere to set off a mine explosion at the face.

Therefore, the electrical fitter must thoroughly examine every piece of flameproof electrical equipment after repair and assembly to be sure that all the bolts are in place and that no joint has a gap greater than permissible.

In the U.S.S.R. detailed requirements for flameproof electrical equipment are laid down in rules and standards for the manufacture of electrical equipment for use in coal and bituminous shale mines issued by the respective State organizations.

Every piece of higher-reliability and flameproof electrical equipment must be examined by the chief mechanic of the mine, or by his deputy every month.

The underground electrical fitter must:

(1) never allow operation of any piece of equipment in a working or at a face cluttered with rock debris, timbers, etc.;

(2) never allow operation of any piece of electrical equipment if some of its cover bolts are missing or poorly pulled tight, when cracks are found in the enclosures, etc., if sealing washers and rubber packing rings are found missing in the cable glands, and when interlocks and protective devices are found to be out of order;

(3) see that all guards around rotating parts on the conveyor drives, winches and other units are in place;

(4) disconnect the supply cables from the starter boxes when units of electrical equipment must be repaired, and post up "Do Not Switch! Men at Work!" notices on the starter boxes;

(5) provide every starter box with a legible inscription to show the piece of equipment it serves;

(6) never allow use of electrical equipment if its insulation is injured or found defective, if the protective earthing connection is missing or the earth leakage protection is out of order or has been cut out.

First Aid to Electric-shock Victims

The very first thing to be done in the case of electric shock is to free the victim from contact with the live circuit by quickly switching the power off or by pulling the victim out of contact with the live parts with rubber gloves and boots on. The rescuer must act swiftly and at the same time exercise care not to receive a shock himself.

If the victim has stopped breathing, he must be taken out into a stream of fresh air without any loss of time, freed of all clothing that will hinder breathing, and given artificial respiration.

Artificial respiration must not be discontinued until a doctor arrives.

In cases of burns by the electric arc, the wound should be covered with sterilized gauze, taking care not to touch the wound with the hands. A call should be sent immediately for the doctor.

Chapter XVII

MAINTENANCE OF MINING EQUIPMENT

Organizing the Work of the Electrical Fitter

Any machine in service is subject to wear, some of its parts lose their initial dimensions, and the materials also deteriorate.

When machines are run at their normal rated loads and are properly maintained, the rate of wear will be slow. However, if a machine is subjected to heavy overloads and is poorly maintained, its wear will be greatly accelerated and may end in a breakdown.

A machine and its parts may thus be subjected to a normal rate of wear and wear due to neglect.

A normal rate of wear is exemplified by the slow wear of the liners in bearings when lubrication is correct.

Such bearings will have to be re-lined only after they have been in service for several months.

An example of wear due to neglect is when the same bearing liners burn out after a very short period of operation (one half to one hour) due to insufficient lubrication.

Wear of machine parts may be due to several causes, namely:

(1) Mechanical wear resulting from friction between moving or rotating parts;

(2) Chemical wear, or what is termed corrosion, due to chemical action upon the parts by moisture, particularly acidic waters, salts, oil, etc.;

(3) Heat wear, or deterioration, resulting from the action of high temperatures on the parts (charring of the winding insulation of electric machines and transformers).

Almost all machines experience vibration during their operation, and the loads on certain of their parts continuously change or alter-

nate. With time fine cracks develop in such parts and gradually grow to a size at which the part breaks, or what is termed a fatigue failure occurs.

If a worn part is not detected at the right moment, an emergency may result during which not only the part but the whole machine will break down.

For example, should a worn bearing in an electric motor not be replaced in time, the rotor will rub against the stator and cause a breakdown because of the subsequent serious damage to the winding and the stator and rotor cores.

The objective of any maintenance is above all to replace or restore the worn parts in a given unit of equipment and also eliminate all other defects detected in it.

Two main forms of maintenance may be distinguished:

(1) Breakdown or corrective maintenance carried out after a unit of equipment has suffered a breakdown;

(2) Planned preventive maintenance by which each machine or piece of equipment is repaired at planned intervals timed to replace worn parts and remedy defects before they can lead to a breakdown.

Proper organization of equipment maintenance obviates the need for breakdown repairs the result of which is always a loss in production time.

Preventive maintenance systems are based on carrying out the examination and repair of equipment according to a fixed schedule.

Daily shift inspections of the equipment are carried out by the respective operators and the shift electrical fitters, both when taking over and during the shift. Minor defects detected by inspection must be immediately remedied.

Whenever the equipment is moved up to a new position, or is newly installed, it is the duty of the respective operators and the shift electrical fitter to examine each machine, mechanism and piece of apparatus.

Maintenance inspections of the equipment are made during the planned maintenance shifts by a team of maintenance electrical fitters from the mine repair shop. The district mechanic and shift electrical fitters also take part in this work. During the inspections defect reports are filled in and the scope of repair work to be done later is estimated.

Maintenance, depending upon the difficulties and the scope of the work involved, may be further classed as:

(1) routine maintenance;

(2) in-service repairs;

(3) overhauls.

Timely and well performed routine and in-service repairs substantially extend overhaul periods. The total service life of equipment is also thereby increased.

Routine maintenance is intended to maintain the equipment in running order and clean. In the main, it does not involve complete disassembly of the units and is performed by the district electrical fitters.

Routine maintenance includes:

(1) washing and cleaning;

(2) renewal of lubricants when necessary;

(3) adjustment and tightening of fastening bolts;

(4) replacement of some accessible parts.

Routine maintenance is carried out during the backshifts according to the preventive maintenance schedule.

In-service repairs are those involving disassembly of the machine or piece of apparatus in the machine shop or a special repair workshop.

In-service repairs include:

(1) replacement of various parts and assemblies such as damaged and worn bearings, friction clutch discs, gears, packings, gaskets, etc.;

(2) remedy of minor defects in machine parts.

During an in-service repair all the operations of routine maintenance should also be performed.

Overhauls of coal mining equipment must be carried out only in special repair workshops or at a mining equipment repair works and should restore the equipment to its full rated capacity.

In addition to the operations of in-service repair, the scope of an overhaul includes the following:

(1) rewinding of electric motors (only if and when necessary);

(2) repair of the frame of any given piece of equipment;

(3) replacement of large parts and assemblies;

(4) restoration of damaged flameproofing enclosures.

When the maintenance of electromechanical equipment in a coal mine is properly organized, the repairs should only involve replacement of worn parts, assemblies and units.

To achieve such maintenance, the mining districts must have an ample supply of spare parts, the condition of the equipment must be closely watched and a planned schedule of replacement of various parts, assemblies and entire pieces of machinery followed out.

Repair work should be mainly routine maintenance and in-service repair with replacement of assemblies, as this considerably reduces the costs of plant overhauls.

Responsibility for the repair of equipment must be shared by the electrical fitters and the coal and entry cutter-loader and cutter

operators and their helpers who are expected to remedy any minor defects and troubles which arise during their shift. They must also take part in the work done by the maintenance and shift electrical fitters. This likewise applies to the workmen who operate other types of machines and mechanisms.

Very close attention must be devoted to the repair of flameproof electrical equipment. In-service repairs and overhauls of such electrical equipment may only be performed in special repair workshops and at mining equipment repair works. In the districts it is only permissible to carry out examinations and routine repairs not entailing the replacement of parts which maintain the equipment flameproof.

Suggested Routine Maintenance for Main Face Equipment

Donbass Cutter-loader

The Drive Unit. Remove the upper cover, examine and replace unsuitable stops and drive pawls.

Adjust claw clutch position.

Check fastening bolts on the upper cover for proper condition and replace bad bolts.

Check the rollers for rotation and the control handles and buttons for proper operation; replace any of the above parts when they have become unfit.

Examine the condition of the cable coupler; replace it if unsuitable.

Electrical Equipment. Check the controllers, replace any fused or burned contacts in them and also any damaged arc chute chambers. Replace controllers completely when they are found to be unfit for further operation. Check the controller operating levers for proper functioning and adjust the back-up spring pressure in the contacts. Tighten the nuts on the controller covers and replace all bad nuts and spring washers. Replace the leather packings when the lubricant is found to be getting into the motor from the drive unit, or into the controller chambers. Replace the jumper cable between the controller and loader motor when it is found to be damaged.

The Cutting Head, Jib, Loader. Replace the jib, cutting chain and gummer. Replace the cutting-bar gear box, the loader flight assembly and the cutting bar unit.

КМП-2 and КМП-3 (KMP-2 and KMP-3) Coalcutters

The Drive End. Remove the cover for inspection of the pulsator spring and slide block.

Check the gear pump for proper operation and for flow of the oil through the pipe lines. Clear clogged lines.

Check the fastening bolts of the upper cover and replace bad ones. Check the rollers for free rotation and the control levers and buttons for correct functioning; replace unfit parts. Check the cable coupler and replace it if it is found to be defective.

Electrical Equipment. Check the controller, clean pitted contacts and replace any damaged arc chute chambers; in cases of extensive damage and wear, replace the controller as a whole. Check the controller operating handle for proper operation, and adjust the contact back-up springs. Tighten the nuts on the controller cover, and replace all bad nuts and spring washers. Replace leather packings when the lubricant is found to be getting into the motor or the controller chamber.

The Cutter Head Assembly. Tighten the fastening bolts on the upper cover and also on the shaft flange. Replace stripped and worn bolts and bad spring washers.

The Jib, Cutting Chain. Replace the chain when the gap between the end of the jib and the saddle exceeds 80 mm (the full travel of the tensioner), or when the number of bad picks exceeds 20 per cent. Replace the jib if the parallel guides have been spread, or wear has increased the dimension to 64 mm.

Scraper-chain Conveyor

Replace the drive shaft assembly together with all the parts mounted on the shaft; replace worn studs and sleeves on the coupling, replace worn troughs, sections of scraper chain containing damaged flights and links, tension-end control levers having worn ratchet teeth, and movable trough pan at the take-up end.

ΠΠM-4 and ΠΠM-5 (PPM-4 and PPM-5) Rock Loaders

Replace unfit loading chain links or the entire chain and the roller chains. Adjust the friction brake for proper operation, including band length and the length of the pull rods attached to the braking levers and pedals. Replace the rubber-belt conveyor idlers and the rubber belt. Lubricate the unit without opening the gear case.

ЭПM-1 and ПMЛ-5 (EPM-1 and PML-5) Rock and Coal Loaders

Replace the pulling chain, the stabilizing ropes and the springs. Check fastening bolts for tightness. Change the lubricant without opening the gear box. Replace burned-out resistors and pitted contacts.

Maintenance of the District Equipment

All the equipment in a given district is operated and maintained under the supervision of the district overman and his assistant, the district mechanic. Every district should have a planned maintenance schedule covering all the equipment employed within its boundaries. On the basis of the separate district schedules, the chief mechanic of the mine sets up and approves a general maintenance schedule for the mine as a whole.

When the equipment is in operation, watch on it is maintained by each shift team, including the shift electrical fitter.

Responsibility for the timely inspections of the equipment used within their district is placed upon the district overman and the district mechanic. The district overman should file a work order with the mine's chief mechanic for repairs of equipment and the necessary spare parts. He is also to see that the given equipment is handed over for repair in time.

The maintenance and replacement intervals are given in Table 25.

Table 25

Equipment	Intervals, months	
	Routine maintenance	Planned replacement
Donbass cutter-loaders	1	8
Coalcutters	1	12
Scraper-chain conveyor drives	2	12
Scraper-chain conveyor drives where shot-fire loading is practiced	1	9
C-153 (S-153) coal loaders	2	12
ППМ-4 and ППМ-5 (PPM-4 and PPM-5) rock loaders	2	12
ПМЛ-5 and ЭПМ-1 (PML-5 and EPM-1) rock loaders	1	12
Hand-held electric drills	—	3
Post-mounted electric drills	1	6
Belt conveyor drives	2.5	12
Booster fans	1	6
District drainage pumps	1	6
Flameproof electric motors of face machinery	1	Replaced together with machine
Flameproof drum-type starters	0.5	As above
Flameproof magnetic starters	1	As above
Lighting transformers	1.5	12

All machines and apparatus handed over by a district for in-service repair or overhaul should be replaced by reserve equipment.

The chief mechanic at every mine should have at his disposal handling teams to move the equipment to and from the required points in the district.

Every mine should have a pit-bottom spare-parts storeroom staffed round-the-clock to permit the districts to obtain a needed part or repair material as soon as the occasion arises.

Such storerooms are obliged to maintain a fixed stock of equipment, tools, spare parts and repair materials in accordance with the inventory list approved and signed by the chief mechanic.

Spare parts for the repair of a machine are issued within the prescribed quota for each district against a procurement form filled in by the district mechanic and initialled by the district overman.

Operators of stationary equipment (pumps, winders, etc.) are obliged to enter all detected defects and abnormalities in the shift log book.

Every item of equipment must be covered by a record card or a log book in which are entered its technical characteristics and ratings, and also data on its operating conditions (duty) and its periods of service at every point of installation.

Also, to have a complete service history, the details of the routine inspections and repairs, as well as the remedy of defects should be entered in the record cards or log books.

When a machine is given an in-service repair or overhaul, the character of the repair work, the names of the repairmen, and what parts have been replaced must be entered in the record card or log book. The latter also serve for keeping a record of all breakdowns and their causes.

In addition to electrical fitters, the operators of equipment in the districts, should also take part in the inspections and routine maintenance.

A qualified operator of a coalcutter, coal or rock loading machine, cutter-loader, or any other machine should not only be able to operate his machine, but also forestall serious troubles and remedy minor defects.

Every shift electrical fitter should have a portable kit containing a set of work tools and also a supply of the most usual expendable items.

The set of tools to be included in the kit is approved by the chief mechanic of the mine for each district in accordance with the equipment employed in the district.

A suggested list of tools for the tool kit of the electrical fitter follows:

One fitter's hammer   One spanner, 5/8″×3/4″
One cape chisel   One box spanner, 3/4″
Two plain chisels   200 grams of friction tape
One drift pin   100 grams of cotton tape
Two screw drivers   One knife
One pair of pliers   One pair of rubber safety gloves
One flat fine-cut file   One pair of calipers
One spanner, 3/8″×1/2″   One folding rule

Large mechanized districts should have an underground workshop. It should have a fitter's workbench equipped with a vice and a cabinet for holding the tools, expendable repair materials and the machine spare parts most frequently needed.

In some cases materials, tools and spares are held in so-called maintenance cars fitted with a cover which can be padlocked.

In his work the underground electrical fitter uses various labour-saving devices such as screw-type pullers for removing couplings from shafts, hand retractable trucks for use in the mounting of equipment, and so on.

Whenever some unit or part in a machine must be dismantled in a gate road, the parts should be laid out on a metal sheet or on clean wooden boards.

Every shift electrical fitter must plan his rounds when on duty in advance so as to avoid any loss of time in going from point to point in his district.

If the face machinery in a district is to operate trouble-free:

(1) Every machine or piece of equipment must be handed over by its operator to the next man going on shift directly at its place of work;

(2) The operators of cutter-loaders, coalcutters, coal and rock loaders, winches, conveyors, etc., must lubricate their equipment in time and in conformance with the lubrication instructions given for each unit;

(3) Every operator in taking over work at the beginning of his shift must look over his machine or piece of equipment to be sure it is in a fit operating condition, ask what troubles were experienced with it during the previous shift, and inform the fitter responsible for the servicing of his unit of equipment or the district mechanic of all defects which have been detected.

The costs of maintenance can be materially reduced by correct operation and care of the equipment, timely repair and thriftiness in the use of the expendable repair items.

Two examples of what can be done in this direction are given below.

The cost of trailing cables is substantially greater and their service life shorter than those of armoured cables of the same size. Therefore

considerable economy may be effected by using armoured instead of trailing cables in gate roadways where only armoured cables can and should be used.
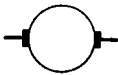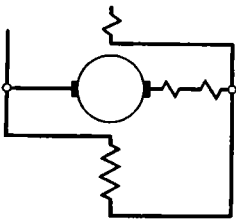
Every mechanized district also requires supplies of many lubricants. If the electrical fitter maintains his equipment properly, immediately remedies any leakage of oil and grease, and sees that the lubricants are properly used, a good deal of the lubricants can be saved.
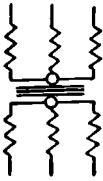
Every machine after a routine repair should be given a trial run in the presence of the district mechanic. This test should be done under load.
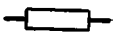
After every such test run the details of the maintenance and the condition of the machine should be recorded in the repairs and inspections log book kept for the main face machinery of the district.
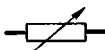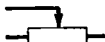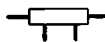
APPENDIX
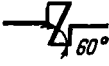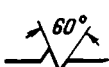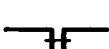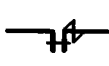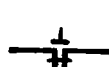
Main Symbols for Electrical Circuit Diagrams According to the U.S.S.R. State Standard

Equipment	Symbol	
	Single-line diagram	Full circuit diagram
Squirrel-cage three-phase induction motor		
Slip ring three-phase induction motor		
D.c. machine		
D.c. compound-wound machine (shunt self-excitation and series windings) with interpoles and compensation windings		
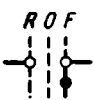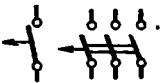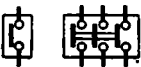
Equipment	Symbol		
	Single-line diagram	Full circuit diagram	Full circuit diagram with windings given
Three-phase transformer with star-star connections			

Circuit element	Symbol
Current transformer with one secondary	
Current transformer with two secondaries	
Semiconductor rectifier	
Mercury-arc grid-controlled rectifier	
Indicating ammeter*	
Recording ammeter*	
Amperehour meter	
Fixed resistor Resistance Ohmic resistance	

Circuit element	Symbol
Impedance	
Adjustable resistor	
Adjustable resistor without break in circuit, with sliding contact	
Resistor with tap-offs	
Coreless inductive reactance	
Iron-cored inductive reactance (choke)	
Reactor	
Fixed capacitor Fixed capacitive reactance	
Adjustable capacitor Adjustable capacitive reactance	
Thermal relay heating element	
Primary or storage cell	
Storage battery	

Circuit element	Symbol
Contactor operating coil	
Voltage (shunt) relay coil	
Current (series) relay coil	
Normally-open contact (N. O.)	
Normally-closed contact (N. C.)	
Normally-open contact (N. O.) with arc extinguishing device	
Normally-closed contact (N. C.) with arc extinguishing device	
Normally-open contact (N. O.) with latching device and electromagnetic release. Release coil is shown separately	
Normally-closed contact (N. C.) with latching device and electromagnetic release. Release coil is to be shown separately	
Normally-open contact (N. O.) with latch and hand resetting	
Normally-closed contact (N. C.) with latch and hand resetting	

Circuit element	Symbol
Normally-open contact (N. O.) with time delay on closing	
Normally-open contact (N. O.) with time delay on opening	
Normally-open contact (N. O.) with time delay on closing and opening	
Normally-closed contact (N. C.) with time delay on opening	
Normally-closed contact (N. C.) with time delay on closing	
Normally-closed contact (N. C.) with time delay on opening and closing	
Auxiliary relay	
Self-return push button with normally-open (N. O.) contact	
Self-return push button with normally-closed (N. C.) contact	
Coupled or lever-arm limit switch with normally-open (N. O.) contact	
Coupled or lever-arm limit switch with normally-closed contact	

Circuit element	Symbol
Master controller, control switch with two positions. Contact makes (●) when turned to right	
Master controller, control switch with two positions. Contact makes (●) when turned to left	
Master controller, control switch with three positions (Forward F, Neutral or Off O and Reverse R). Normally-open contact (N. O.) makes (●) when turned to right and stays open when turned to left	R O F

Circuit device	Symbol
	Single-line diagram and full circuit diagram
Automatic three-pole load-breaking switch, air circuit breaker	
Three-pole circuit breaker (with arc extinguishing in oil, by oil jet, by water or air blast, etc.)	
Three-pole knife switch Three-pole isolator	
Three-pole changeover switch Three-pole changeover isolator	
Plug and socket connection	Plug Socket
Plug-in connection in withdrawable type devices and switchgear	

Circuit element	Symbol
Electric bell	
Siren, horn, howler	
Signal (indicating) lamp**	
Spark-gap	
Fuse	
Cable termination	
Lighting lamp	

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* In the case of instruments for other quantities the respective letter marking is used; V for voltmeter, W for wattmeter, etc.
** The respective colour of the signal or the condition it indicates should be denoted above the general symbol by the first letter of the given colour or condition; for example, W when white, E for emergency, etc.

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